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The IM& E Journal 1457, Fishery Tank Road, Chintamaniswar, Laxmisagarpatna, Bhubaneswar - 751006, Odisha
Mobile: +919861008387, **Mail:** indianminingjournal61@hotmail.com
Branch office:Near TV Tower, JODA, Dt. Keonjhar 758034
Phone: 06767-273173
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Message

I am happy to know that Indian Mines and Engineering Journal has completed its 60 years of successful journey and fulfilling the requirement and demand of Geo Scientist across the world with its publications.

The Journal has contributed exceptionally in serving the nation for development of R&D and having reached to the industrialists, academicians, scientists and technocrats.

I congratulate the Chief Editor Prof. G.K. Pradhan and his team for their dedicated effort throughout who are giving their best to make the event a grand success.

I wish him all the best !

T.N.Singh
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Influence of Lockdown Amid COVID-19 Pandemic on Mining Activities in India

Vivek K Himanshu* Shubham Kumar* Ashish K Vishwakarma* Murari P Roy*

ABSTRACT

COVID-19 pandemic affected the lives of millions of people around the globe. The mineral production pace was also affected during the pandemic. Some of the minerals have shown higher reduction in production, while some of them had mild impacts. The nature of impact was mainly dependent on the government obligations during the pandemic. However, some of the highly mechanised mines have shown very fast growth in the production rate after the relaxation in lockdown. This paper presents a review on the nature and severity of impacts on mineral production pace during the pandemic.

INTRODUCTION

The mineral and coal production is the backbone of the economy of a country. It contributes directly and indirectly to the infrastructural sectors, energy sectors and strategic sectors. COVID-19 pandemic has influenced the major industrial productions around the globe including mining sector. The influence of the pandemic on mining activities related with different minerals has been reviewed in this paper. The production and safety related data has been collected for this purpose from the internet. The mineral production data was collected from the website of Indian Bureau of mines. The data pertaining to the mining accidents has been collected from the website of ENVIS centre on environmental problems in mining.

The review reveals that the nature of influence on different mineral was different during the pandemic. The variations in influence was due to the government obligations on different sectors, the nature of working by the organisations involved with the concerned mineral production and location of the mineral deposition. The spreading rate of pandemic and government obligations during different periods has also been reviewed for this purpose.

COVID-19 PANDEMIC AND GOVERNMENT OBLIGATIONS

The very first covid-19 case was reported in Wuhan, china during the month of December 2019 (Wikipedia.org). The World Health Organisation (WHO) declared the covid-19 as public health Emergency of International Concern on 30th January 2020 and the pandemic on 11th march 2020. In India, first COVID-19 infection was reported in Kerala on 27th January 2020. The Government of India Ordered nationwide lockdown for 21 days on the evening of 24th March 2020. The summary of government obligations during different phases of lockdowns are given in Table 1.

REVIEW OF MINERAL PRODUCTION DURING PANDEMIC

An overview of total mineral production during January to August of 2019 and 2020 is shown in Figure 1. The study of the trend of coal and mineral production shows that almost all the minerals except Lead & Zinc had declining trend of production during 2020. The coal production trend had no severe impact. There was minor 1.5% decline in the coal production during January-August 2020 as compared to January-August 2019. The comparison of the monthly production trend of coal (Figure 2) shows that a maximum 14% decline in the coal production was in the month of April. The production has started ramping up from August. The coal was considered as the essential raw material for power generation during the lockdown. So, there was almost no impact of COVID-19 on coal production.

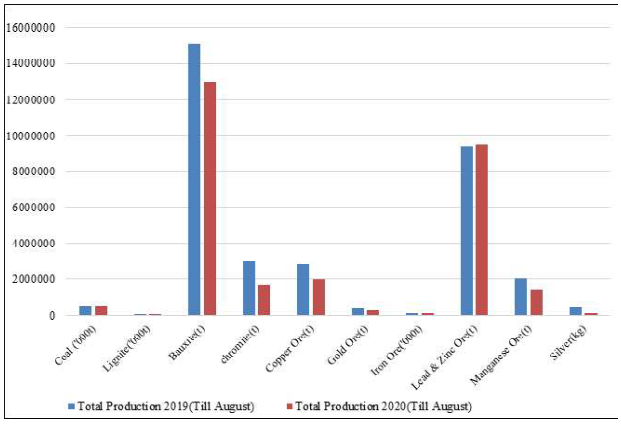

Figure 1: Comparison of total coal and mineral production during January to August of 2019 and 2020.

**CSIR-Central Institute of Mining and Fuel Research, Dhanbad-826001, India*

Table 1. Summary of government obligations during different phases of lockdowns

Lockdown	Period	Obligations and Relaxation by Government
Lockdown Phase 1	March 25- April 14, 2020	Nationwide lockdown, movement was allowed only for essential services (vegetables, food items and medicines)
Lockdown Phase 2	April 15- May 03, 2020	certain relaxations allowing agricultural businesses, including dairy, aquaculture, and plantations, as well as shops selling farming supplies, to open. Public works programmes were also allowed to reopen with instructions to maintain social distancing.
Lockdown Phase 3	May 04-17, 2020	The country was splited into 3 zones: red zones (130 districts), orange zones (284 districts), and green zones (320 districts). Red zones were those with high coronavirus cases and a high doubling rate, orange zones were those with comparatively fewer cases than red zone and green zones were those without any cases in the past 21 days. Normal movement was permitted in green zones with buses limited to 50% capacity. Only private and hired vehicles but no public transportation was allowed in orange zones. The red zones were remaining under lockdown. The zone classification was revised once a week
Lockdown Phase 4	May 18-31, 2020	Unlike the previous extensions, states were given a larger say in the demarcation of Green, Orange and Red zones and the implementation roadmap. Red zones were further divided into containment and buffer zones. The local bodies were given the authority to demarcate containment and buffer zones
Unlock-1	June 01-30, 2020	Lockdown was only in the containment zone. From 8 June, shopping malls, religious places, hotels, and restaurants were allowed to open but still large gathering is prohibited. Night curfew was in effect from 9 P.M. to 5 A.M. Large gathering was still prohibited. The state government was allowed to impose suitable restriction.
Unlock-2	July 01-31, 2020	Lockdown were imposed to only contaminated zone. The state government had the power to impose suitable restrictions. Interstate travel, limited international travel and shops with more than five persons at a time were permitted. Night curfews were in effect from 10 PM. to 5 AM. in all areas.
Unlock-3	August 01-31, 2020	Night curfew removed. Gymnasiums, yoga centres reopens from 5th August and interstate travel and transport were permitted. Maharashtra and Tamil Nadu imposed lockdown for whole month, while West-Bengal imposed lockdown twice in a week. From 30th august Delhi Metro stared two routes.

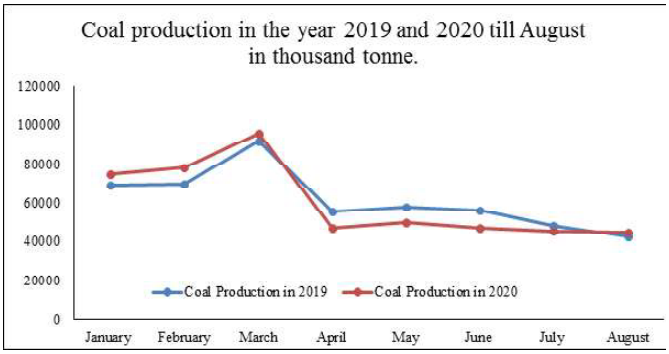

Figure 2: Comparison of monthly coal production during 2019 and 2020.

The lignite is a sedimentary rock formed from compressed peat. It is considered as lowest rank coal due to its low heat content. The dominant lignite producing state in India is Tamil Nadu. The lignite posed a declining production of around 11.5% due to lockdown. The declining trend (Figure 3) started from the month of March and continued till August with slight ramp up. The review of yearly production of lignite (Figure 4) reveals that the lignite production pace differs across the years. Hence, the main reason of decline is due to the operational constraints of mines.

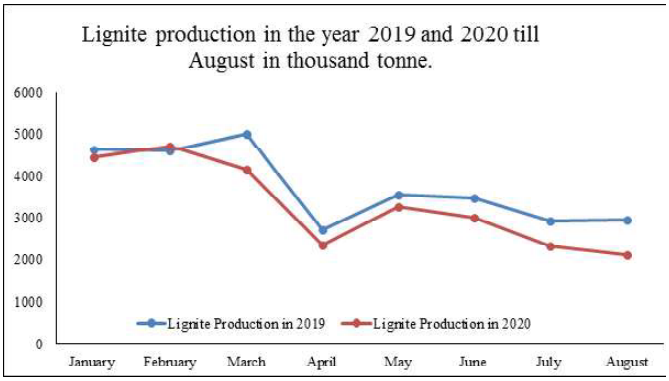

Figure 3: Lignite production during 2019 and 2020

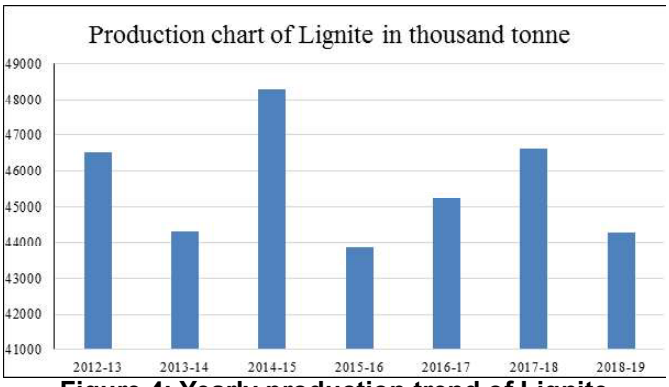

Figure 4: Yearly production trend of Lignite

Bauxite is the raw material for Aluminium. Odisha is the maximum bauxite producing state in India. It has shown around 14% decline in the production during lockdown. Around 50% decline in production was observed during April. The production pace was retained during July and August (Figure 5).

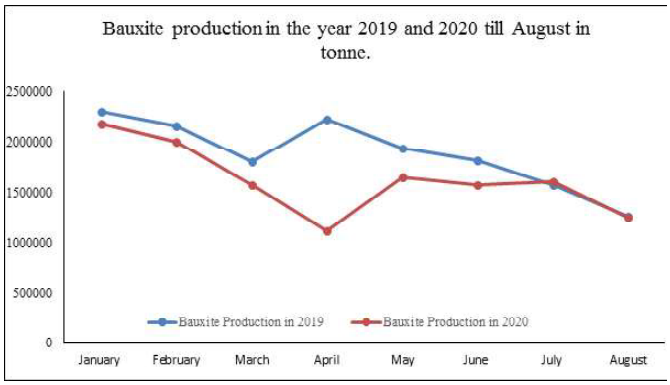

Figure 5: Comparison of Bauxite production during 2019 and 2020.

The maximum impact of COVID-19 has been seen on production of chromite mineral. Chromite is a crystalline mineral used dominantly in steel industries. More than 96% of chromite deposition is in Odisha. The decline in chromite production due to lockdown was more than 45%. The decline trend was all along from April onwards (Figure 6). The review of yearly production of chromite shows that it was in increasing trend over the years (Figure 7). This means that the production was impacted due to the pandemic.

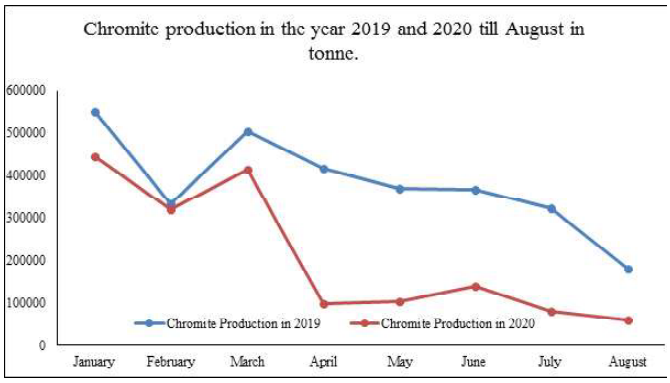

Figure 6: Monthly production of Chromite during 2019 and 2020.

The production of copper ore was also dominantly impacted due to the pandemic. Copper is mainly used in electrical equipment for wirings and motors. The mining of copper ore deposits takes place in Madhya Pradesh, Rajsthan and Jharkhand. The decline in copper production

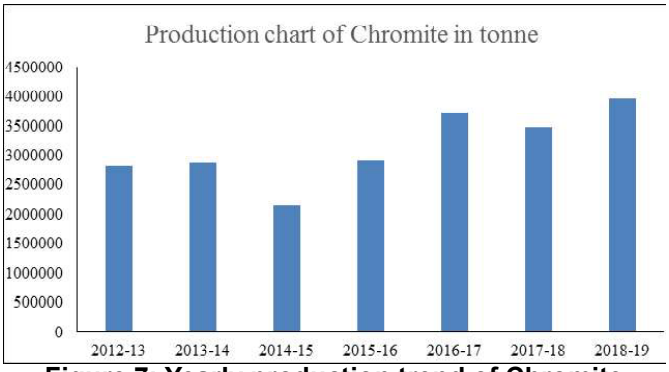

Figure 7: Yearly production trend of Chromite

was more than 30%. The maximum decline of 45% and 70% was seen during March and April (Figure 8). The mineral production has started gaining pace onwards, but it was in declining trend with respect to previous year until August 2020.

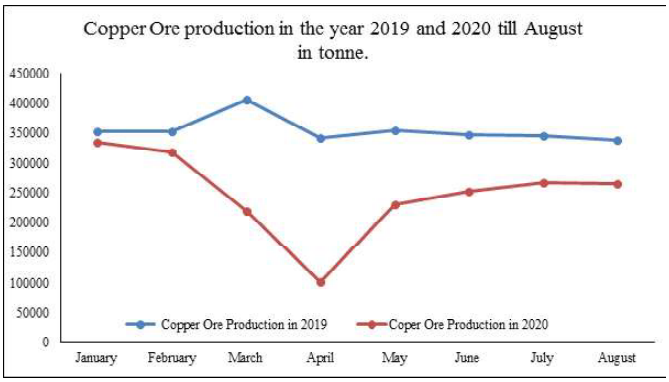

Figure 8: Copper ore production trend during 2019 and 2020

The production of manganese ore has also shown a sharp decline of about 30% during the pandemic. Manganese is used mainly in electronics industries. The dominant manganese producing state are Madhya Pradesh, Maharashtra and Odisha.

The overall decline in the production of Iron ore was relatively low. It was near about 15%. The comparison of monthly production statistics of Iron ore (Figure 9) shows that the production pace of Iron ore during the initial months of 2020 before pandemic was very high. The production during January-March of 2020 was more than 17% for the same period during 2019. Due to the impact of pandemic, the iron ore production reduced sharply and it showed a decline at the end.

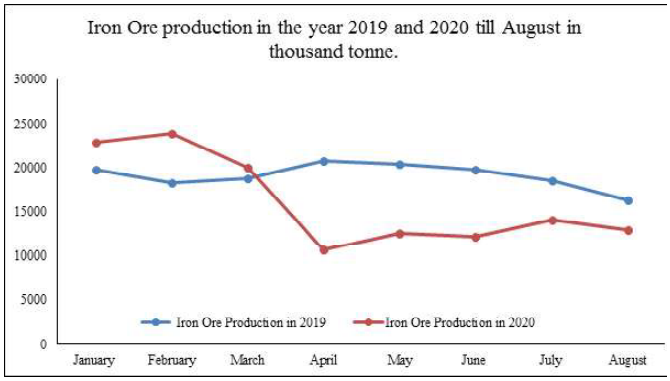

Figure 9: Trend of Iron ore production during 2019 and 2020.

The production pace of Lead-Zinc ore extraction was almost same with a little enhancement of about 1%. It is the only mineral which showed the enhancement characteristics despite of pandemic. Lead-zinc ore deposition is present in Rajsthan state in India. The mineral has shown decline in production during the complete lockdown period. The maximum decline in production was around 41% during April (Figure 10). However, the production rate has ramped up fast in the unlock period. The Lead-Zinc mines are most mechanised mines in India. The faster ramping up of the production was due to the mechanical mining and less dependency on manpower.

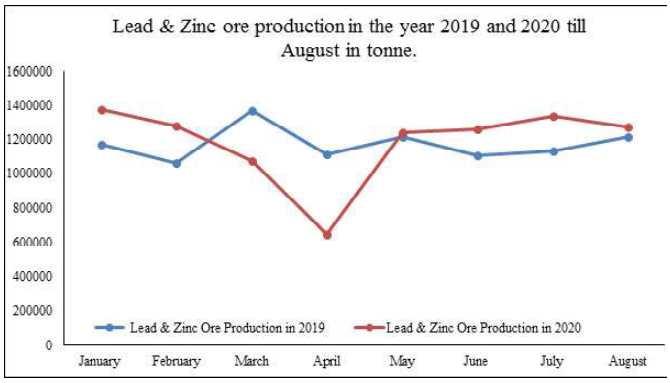

Figure 10. Comparison of Lead-Zinc ore production during 2019 and 2020.

SAFETY INCIDENCES DURING PANDEMIC

The review of the safety incidents of the mine reveals that the incidences were reduced during the pandemic. The reduction was mainly due to lesser involvement of human resources in mining activity. The summary of safety incidents in Indian mines during 2019 and 2020 is shown in Table 2 and Table 3 respectively. The table shows that

total 10 mining induced accidents were recorded during 2020 by ENVIS centre as compared to 18 mining induced accidents during 2019.

Table 2. Mining induced accidents during 2019

SL. No.	Month	Killed	Injured	Cause
1	January	00	00	
2	February	04	00	1.Toppling of Tripper 2.The operator fell down between the blade and crawler of the dozer 3.Due to Electricity 4.Run over by the LHD
3	March	02	01	1. Run over by grader 2. Blasting
4	April	02	00	Falling of the sides of six overlying benches 2. Lightning
5	May	02	01	1. Due to Electricity 2. Run over by the loaded coal tipper 3. Bucket for loading blasted waste rock
6	June	02	00	1. Run Over by Grader 2. Due to Electricity
7	July	04	00	Rib failed
8	August	00	00	
9	September	01	00	Fell down in next lower bench
10	October	00	01	Electrically operated LHD
11	November	00	00	
12	December	01	00	Haulage Rope

Table 3. Summary of mining induced accidents in India during 2020

SL. No.	Month	Killed	Injured	Cause
1	January	02	01	1.Falling a mass of stone 2.Blasting
2	May	04	00	1.Transportation machinery (Conveyors). 2. Run over by a dumper. 3. Wheeled Trackless (Tipper) 4. Accident with dumper.
3	June	01	00	Explosive
4	July	01	00	Drowning in water.
5	December	02	02	Flooding of underground Mines.

CONCLUSIONS

The review of the mineral production during the period of pandemic shows that, it was impacted due to lockdown. The impact on essential mining of coal was comparatively less due to its role in power generation. The minerals used as raw material in metalliferous sand steel plants were having high impacts as they were not considered essential during lockdown. Some of the mineral excavation has recovered the decline sharply during the period of unlock. The recovery was sharper in the case of mechanised mining. The mining induced accidents were also reduced during the period of pandemic. Total 10 accidents were recorded during 2020 by ENVIS centre as compared to 18 mining induced accidents during 2019.

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Efficient Productivity with Preventive Measures for Occupational Hazard in Stone Quarry

Suraj Dev singh* Sandeep Prasad**

ABSTRACT

In this modern era the Rapid growth in construction industry as well as the Urbanization of the rural areas, the demand of aggregate has immensely boosted and the production of the raw material is not much sufficiently provided to the crushing plant to meet the actual requirement or the demand, having the various reasons i.e. deterioration of the stone quality, lack of Geo- technical knowledge, method of work for the stone excavation, illegal stone mining which directly promote the unsafe working condition for the workers and facing the many health problems due to heavy inhalation of exposure to the dust and Government clearance -To improve the productivity the executive or Owner has to fulfill all the silent features of the project, written on the executive summary of pre-feasibility report for smooth running the stone quarry.

As we know that Mining industry is a hazardous industry and the activities done in stone quarry, Which is associated with the various respiratory diseases such as Pneumoconiosis caused due to accumulation of dust in the lungs and the functional impairment and the radio graphic abnormalities of the lungs and maximizing environmental impact and other occupational hazard during drilling and blasting and Transportation of stone from the quarry. To minimize the occupational hazard various preventive majors have to be taken such as now-a-days non- explosive demolition agents are used to eliminate the accidents occurring during drilling and blasting and It is a primary cause of the stone quarry's production being reduced, and it has occurred owing to a lack of geotechnical understanding or to put it another way, a lack of skilled workers.

Now a substitute has been introduced to the stone Quarry i.e. cracking powder which is environmentally friendly and safe to use with effective productivity.

Keywords: Pre-feasibility, Pneumoconiosis, Cracking powder.

INTRODUCTION

The technique of extracting stones from natural rock surfaces is known as quarrying. The site selection and quarrying procedures utilized for construction projects are discussed. Stone quarrying is not the same as mine quarrying. Quarrying is done on the exposed surface of natural rocks, whereas mining is done underneath.

As a result, quarry stones are employed for a variety of engineering purposes. Stone quarrying is typically done in hilly places where there is a big supply of stone.

The quarry should be chosen based on the following criteria.

1. The site should not be close to human habitations and wild life sanctuary. Because the vibrations caused by blasting in the site may affect permanent infrastructure and wild life animals.
2. The mode of transportation system should be provided to carry the raw materials or stone boulders.
3. Near the quarry site, there should be a clean water source so that the workers deployed can get the fresh water.
4. Stone of good quality and quantity should be easily accessible.
5. It is necessary to read the site's geological information.

Factors to Consider When Quarrying Stones

1. Before beginning stone quarrying, some critical decisions must be made after the location has been chosen. These are the following:
2. Cracks and fissures in the rock surface should be thoroughly examined. These could cause planes in the stones, which could lead them to split. Quarrying of stone will then be simple, rapid, and cost-effective. And using a chisel, axes, and hammer, it is a primitive technique for separating a block of rock with pre-existing fractures and fissures from the intact solid mass.
3. The machines should be examined to ensure the capacity so that the material can be easily mucked from the site and transported to the crushing plant.

**M.Tech Student **Assistant Professor, Department of Mining Engineering, AKS University, Satna*

4. The removal or loading of stones should be done with caution, as there is a risk of landslides or slips, which can result in minor as well as serious injury to workers which will reduce the productivity of the mine.

Problems associated with the stone quarry

Vibrations, soil degradation, land subsidence and landslides, water pollution, occupational noise pollution, and air pollution, all of which are associated with quarrying activities which will result in health issues and biodiversity loss.

Quarrying operations have the potential to disrupt pre-existing ecosystems as well as hydrogeological and hydrological regimes. This negative impact of stone quarrying causes property damage, groundwater depletion, loss of fertile topsoil, forest degradation, aquatic biodiversity decline, and public health issues.

Through the study of various aspects of stone quarrying, it has been observed that the activities carried out in a stone quarry have a direct impact on human health, such as the respiratory system due to heavy inhalation exposure to dust and other occupational accidents that result in minor and serious bodily injury, which reduces the mine's productivity.

The case study has been obtained by the mines that uses chemical cracking powder, and it is observed that it is environmentally friendly and safe to use with effective productivity.

When it is compared to blasting method, environmental impacts such as ground vibration and risk of occupational hazard due to fly Rock and also generation of dust is more through using of explosive.

METHODOLOGY

Non-Explosive Demolition Powder i.e. Use of stone cracking powder (Figure 1)
It has been observed that the use of stone cracking power in a mines are eco-friendly with nature and safe to use or having less occupational hazard such as problems associated with ground vibration and respiratory problems due to heavy inhalation exposure to the dust.

The mines from where the case study was obtained uses the "crackmax" and it is a non-toxic powder made up of calcium, silica, and aluminum oxides (Figure 2).

Crack Powder is a powder made up of an inorganic compound mostly made up of a certain type of silicate and an organic ingredient. There are no dangerous ingredients in Crack Powder. It offers the most technically appropriate and cost-effective solution in the field. It is safer method compare to blasting of rock because when surrounding structures are being demolished by blasting. Shock waves caused by explosives are more hazardous because of use of Explosives.

The techniques to utilize the cracking powder i.e. 5kg of cracking powder is mixed properly with 1.5 liter of water and make the slurry , then it is filled to hole of 5ft (i.e. depend on the strength of the rock) and then move away to a safe place while the chemicals work to split out the rock block.

Figure 1: Filling of cracking power in drilled holes

Figure 2: Rock fragmentation by cracking power

TAMPING METHOD

- In this method after placing the charge in the hole, a greased priming needle, projecting a little outside the

hole, is place in the hole which is filled with the clay or inert material or stone dust in layers tamped sufficiently with a braced tamping rod.

- The priming needle should be kept on rotating (so that it can be easily removed) while tamping is going on.
- The priming needle is then removed out and 60-70% of space created by withdrawal of needle is filled with explosive.

By using this method the amount of consumption of explosive will be less, it will be cost effective as well as it reduces the ground vibration so that, it will result to minimize the industrial hazard.

CONCLUSION

It has been observed that the "crackmax" is the most powerful stone cracking powder. A single explosion can dislodge more than three tons of obstructions in milliseconds. And it is the most relevant factor determining for the selection of demolition agent as compare to blasting method.

It has been seen that after dislodge of the rock lumps from the intact rock mass and which is compared with blasting method. It found that cracking powder is environmentally friendly and safe to use with effective productivity or can say cost effective method.
If we use the blasting technique, have to follow the general precautions so that occupational hazard might be reduced or minimized .the hazard is directly proposal to the mine productivity.

- Blasting hours should be made public and siren should be warn the workmen and nearby public timely to retire to a safe distance or we have to alert the people that blasting operation is going on.
- The danger zone should be marked with red flags the danger zone should be defined by an area of about 200 to 300 meter radius.
- First aid or first aid kit should be readily available in case of any emergency.
- Explosive should be stored and handle carefully and detonators and explosive should not be kept together.
- Cartridges should be handled with rubber or polythene gloves.
- Maximum boreholes are exploded at a time and that also successively.

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Uncontrollable and Controllable Parameters of Rock Blasting - A Review

Ranjit K Paswan* Vivek Priyadarshi* Sourav Kushwaha* Vishal Sagar Rana* Murari P Roy* Pradeep K Singh*

ABSTRACT

The rock mass discontinuities play important role in blasting operation and influence the results of blasting in many ways such as overbreak, underbreak, boulder formation, toe formation etc. the most common uncontrollable parameter in blasting operations are the mappable un-controllable parameter such as joints, bedding planes, fracture, fold and fault etc. These inherent property of rock masses increase the total project cost and create difficulties in blasting operation. Rock mass structural discontinuities have remarkable effect on the physio-mechanical property of the rock. The blasting process can be described reasonably well in words but the blasting science have not yet been able to model the detailed mechanisms involved or describe the influence of particular rock mass properties on field blasting performance in other than qualitative terms. Experience has shown that the excessive blast damage to the perimeters of underground and surface excavations due to presence of discontinuities can be controlled in an appreciable manner by careful blast design keeping in view the discontinuities present in the rock masses. For the optimum result like desired fragmentation with control blasting achieved in blasting operation over range of inherent property of rock mass condition, it is important to understand the role of geological discontinuities of rock mass. Apart from discontinuities, the desired fragmentation of a particular blast can be influenced by blast design parameters. Six blasts were conducted and fragmentation analyses were carried out with respect to blast design. This paper reviews the influence of controllable and uncontrollable parameters on blasting results and the remedial measures proposed by different researchers.

INTRODUCTION

In mining and civil project there is serious concern about the cost-effective mining with safety and productivity. The role of geological discontinuities is important to understand to achieve optimum blasting results. Rock mass are formed in different stress condition and due to these stress conditions geological discontinuities are created in principal stress directions. It is very rare to find rock free from any geological discontinuities such as joints, fracture, fault, bedding planes joints etc. (Singh 2005, Paswan et al. 2017). In blasting operation during the explosion after loading of blast holes seismic waves are produce is similar to the earthquake but seismic waves are produced in blasting operation is very high frequency and very low amplitude which not damages the surface structures. The fragmentation of rock mass depends on the energy produces by the explosive used in blasting operation (Ramulu et al. 2009). In underground mines and tunnels jointed rock masses is more severe and vulnerable for safe excavation (Singh & Xavier, 2005). Many researchers in the world documented the blast induced damages on underground openings (Langefors & Kihlstrom 1963, Hendron 1977, Holmberg 1993, Singh 1993, Chakraborty et al. 1998). In an underground openings during the blasting operation rock masses damages along in the direction of their joint orientation (Cunningham & Goetzsche, 1996). Lewandowski et al. (1996) observed largest overbreak for the joint orientations of 45° and minimum overbreak for parallel and perpendicular joint's orientations and similar results were also obtained by Singh & Xavier (2005).

The majority of researchers and scientists (e.g., Rustan et al. 1983, Fourney et al. 1993) derive from their investi-gations that there is difference between stress wave propagation microcrack orientation, microcrack length and the crushed zone around the blast hole for open or filled joints. During the explosion the first inherent fracture open and dominate for further fracturing (Singh and Sastry, 1987). In joint attenuation of stress wave depends on the angle of incidence of stress waves with joint face (Lewandowski et al.1996). Worsey and Qu et al. (1987) and Whittaker et al. (1992) derive from model tests that closed joints have a di-minishing influence whereas open joints affect the stress wave transmission.

Breakage of rockmass is controlled by the coupling of the explosive on the wall of the blast hole in the confining pressure. Sharma (1994) provides a useful description

** CSIR-Central Institute of Mining and Fuel Research, Dhanbad-826001, Jharkhand*

of the blasting process as a back ground to modelling explosive-rock interaction (Figure 1). This model of the blasting process identifies several rock mass properties important to blasting performance. These includes: rock mass stiffness which control the distortion of blast hole and pressure developed in blast hole. Dynamic compressive strength of the rock mass. Attenuation property of rock mass, dynamic tensile strength of the rock mass which is responsible for new fracture development, In-situ fracture present in rock control block size through which seismic wave attenuation and migration of explosion gases and density of the rock mass.

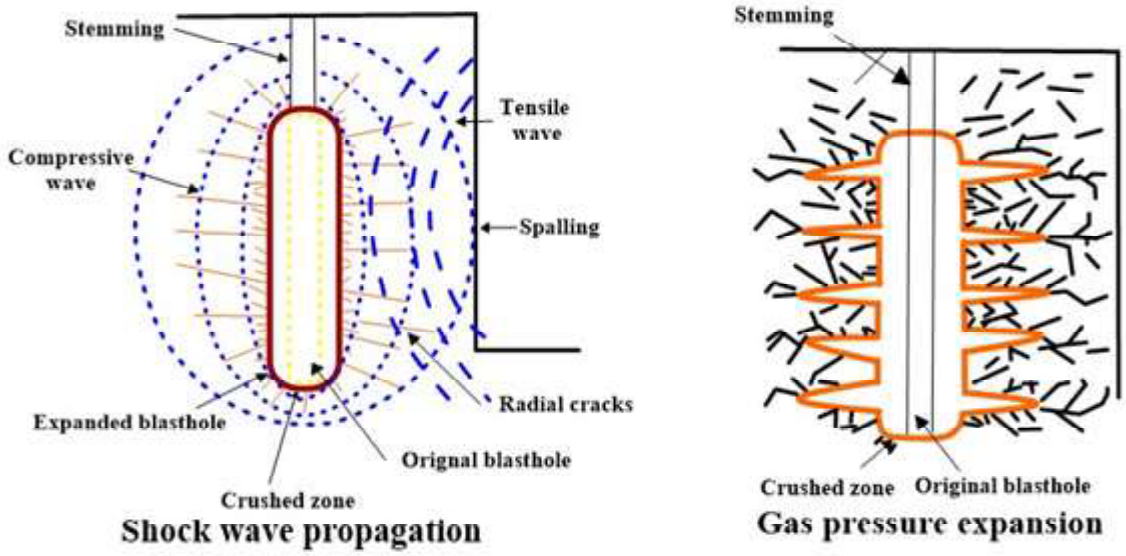


Figure 1. Shock wave propagation and gas pressure expansion in a rock mass (Sharma, 1994)

Controllable	Non-controllable
❖ Basic design parameters : ▪ bench height & hole depth ▪ burden ▪ spacing ▪ sub-grade ▪ explosive column length & ▪ stemming length etc.	❖ Local geology
❖ Mode of initiation of explosives in a blast hole and blast delay pattern initiation	❖ Rock characteristics
❖ Blast face condition	❖ Distance between explosion site and structure
❖ Air decking/decoupling	
❖ Type of explosives	
❖ Effective quantity of explosives in a delay	
❖ Total quantity of explosive in a blasting round	
❖ Delay timing between two detonations	

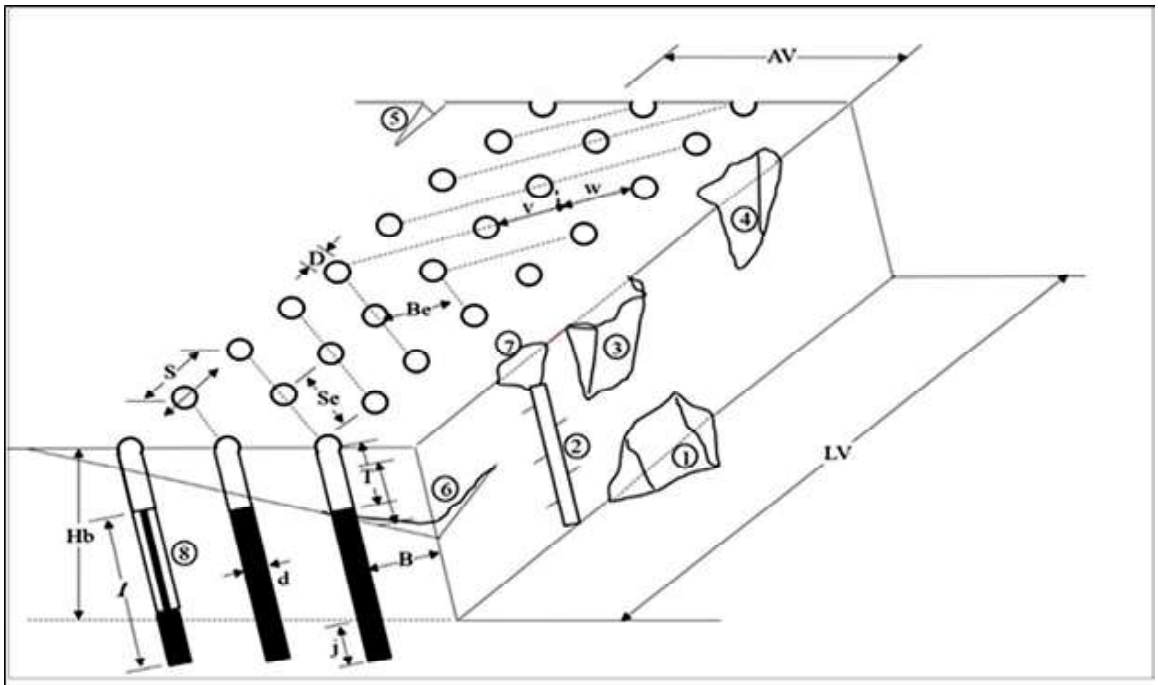
CONTROLLABLE PARAMETERS OF BLASTING

The parameters of blasting operation which are in control of blast operating person have a significant impact in minimizing the adverse impact of blasting such as ground vibration, flyrock, air-overpressure etc. Jemino et al., (1995) classified the controllable parameters of designing a blast in fallowing points:

A. Geometric
B. Physicochemical or pertaining to explosives
C. Time (Delay timing and initiation sequence).

Bench height (H_b) and blasthole depth (L)

This is the vertical distance between the floor- and the top level of a bench **(Figure 2)**. The stiffness (ratio of the height of bench to burden, H_b/B) of the rock located in front of the blastholes have a significant influence on the result of blasting. Ash (1977) states that the optimum ratio is H_b/B ~3. If H_b/B = 1, the fragments will be large, with overbreakband toe problems. With H_b/B = 2, these problems arebattenuated and are completely eliminated when H_b/B ~ 3. In the case of larger bench heights, there is a possibility of blasthole deviation which will not only affect rock fragmentation but will also increase the risk of generating higher ground vibrations, flyrock, and overbreak as the drilling pattern B × S will not remain constant in the different levels of the blasthole.



Hb = Bench height, D = Blasthole diameter, L = Drilled length of blasthole, d = Diameter of the explosive charge B = Burden, S = Spacing, LV = Length of the blast area, AV = Width of the blast area, Be = Effective burden distance Se = Effective blasthole spacing, T = Stemming length, J = Sub-drilling, I= Charge length, e= Angle subtended at the centre of the shot by charge fired in the same delay, v/w = Degree of the balance of the effective staggered blasthole, 1 = Toe, 2 = Half cast, 3 = Overhang, 4 = Overbreak or backbreak, 5 = Tension crack, 6 = Cut-offs, 7 = Fizzle crater, 8 = Decoupled charge

Figure 2. Design parameters of the blast (modified after Jemino et al., 1995)

The bigger bench height for good fragmentation as well as the economic exploitation of the mineral is the common trend and to achieve this larger blastholes are essential. The blasthole depths have a profound influence on blast induced ground vibration as in larger blasthole depths the generation of vibration is more because of the concentration of explosive at depth. It was reported by Kuzmenko et al., (1993) that when bench height is increased by 3.5 to 4 times the burden, the displacement velocity increases by 30%. They further explained that

this enhanced intensity is due to the increase in the total weight of explosive in blasthole.

Blasthole diameter (D)

The diameter of the blasthole into which the explosive is to be placed has a very crucial role in overall blast performance. Before drilling the blasthole, there are few factors which need to be taken in to consideration. They includes properties of the rockmass to be blasted, degree of fragmentation required, height of bench and configuration of charges, costs of drilling and blasting and capacity of the loading equipment. The smaller blasthole diameter (D) involves high costs of drilling, priming, and initiation. At the same instant charging, stemming and connection take a lot of time and labour. The only advantage of smaller 'D' is a lower powder factor because of the most optimum distribution of the explosive. The increase in 'D' has various advantages. They comprises higher detonation velocity which gives more stability and is less influenced by external conditions, lower overall costs of drilling and blasting, Possible mechanization of the explosive charge, higher drilling productivity (m^3 blasting/ml drilled), and increase in shovel yield as a consequence of fewer low productivity zones. If 'D' is increased, while keeping fragmentation constant, it will be necessary to increase the powder factor as the charges are not evenly distributed in the rockmass. The stemming length T increases with the drilling diameter and the collar of blasthole could become a potential source of boulder formation.

Blasthole inclination

The inclined drillings of blasthole have numerous advantages as well as a few disadvantages. Usually, in large open pit mines where rotary rigs are used, vertical blastholes seem to be the tendency; however, with rotary percussive drilling equipment, the blastholes are inclined. The benefits of inclined drilling include Better fragmentation, displacement, and swelling of the muckpile, as the burden B value is kept more uniform along the length of the blasthole and the angle of the projection direction of the shot increases, (**Figure 3**), less probability of misfire caused by cut-off from burden movement **(Figure 3)**, Smoother and sounder slope in the newly created benches, higher productivity of front end loaders due to more swelling and lower height of the muckpile, less sub-drilling and better use of the explosive energy, with the consequent lower vibration level, lower powder factor as the shock wave is reflected more efficiently in the bench toe and the possibility of increasing burden size with less risk of toe appearance, (**Figure 4**) (Jimeno et al., 1995).

Besides these, there are few disadvantages also, which includes increased deviation when drilling long blastholes, increased drilling length, difficulty in the positioning of the drills and in collaring operations.

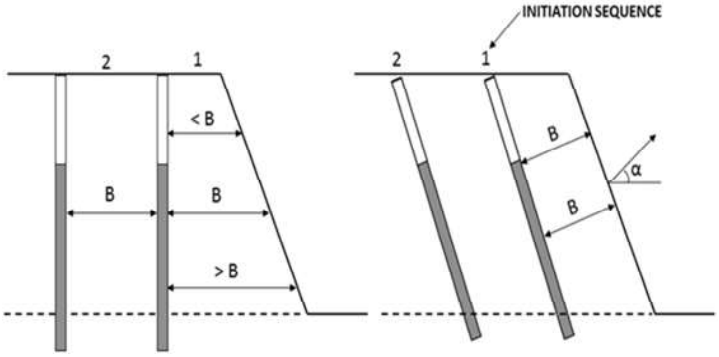


Figure 3. Inclined drilling versus vertical drilling (Modified after Jemino et al., 1995)

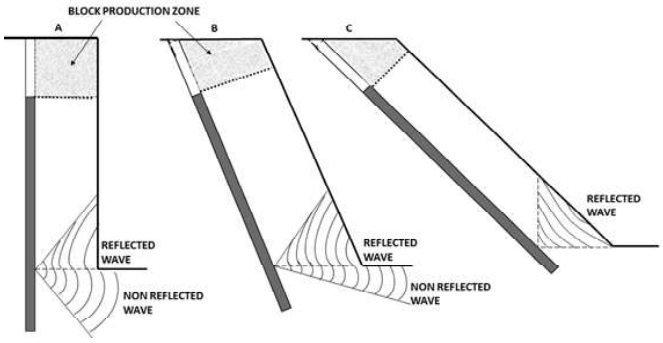


Figure 4. Benefits of inclined holes (Modified after Jemino et al., 1995).

Basic dimensional factors

The four interdependent factors employed for single charge blasting design are burden, stemming, sub-drilling and charge length. For multiple charge, spacing is also included (Ash, 1973).

Burden

Burden (B) is the most critical design parameter as it has the greatest influence on fragmentation and ground vibration (Allsman, 1960; Langefors and Kihlstrom, 1963).

The determination of optimum burden is very critical from ground vibration point of view as the extra burden may increase the ground vibration. The reason being that the explosive energy is insufficient to break the burden, subsequently, converted into ground vibration. On the contrary the in sufficient burden leads to extensive flyrock and air blast rather than increased ground vibration (Hagan and Kennedy, 1977).

Rustan, (1992) critically reviewed all the existing formulae for burden and suggested the use of a power formula, one for opencast mines and one for underground mines:
For opencast mines:

$B_{p1} = 18.1 \cdot d^{0.689}$ +52 % expected maximum value …. 3.8

-37 % expected minimum value …. 3.9

The above formula is valid for blasthole diameter from 89 to 381 mm.

For underground mines:
+40 % expected maximum value …. 3.10
-25 % expected minimum value …. 3.11

The above formula is valid for blasthole diameters from 48 to 165 mm.

Numerous formulae have been suggested to calculate the burden, which takes into account one or more of the indicated parameters. However, all the values of burden lie in the range of 25 to 40 times of the blasthole diameter, depending fundamentally upon the properties of the rockmass (**Figure 5**).

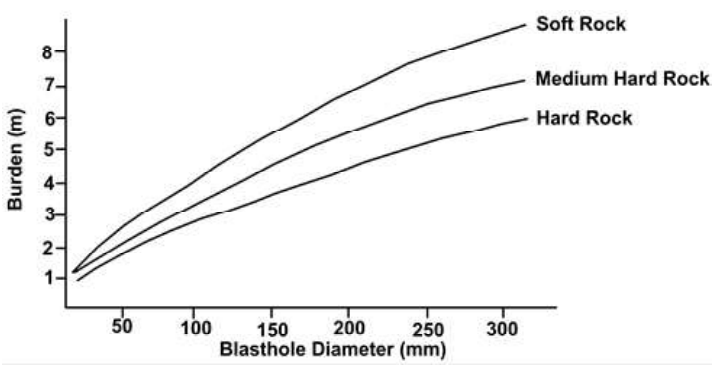


Figure 5. The relation between maximum burden and blasthole diameter (Jimeno et al., 1995).

Spacing

Bhandari, (1977) has described in model blasting as well as from his field data that spacing to burden ratio of greater than one, increase the fragmentation, which in turn means less vibration. This ratio may be kept lower than one only in the case of presence of weak joints or irregular backbreaks. The investigators have proposed different spacing/burden (S/B) ratio (**Table 2**).

Table 2. Spacing/burden (S/B) ratio suggested by different investigators.

Investigators	S/B ratio
Hino, (1959); Speath, (1960); Fraenkel, (1962)	1.3-1.4 for simultaneous firing
Langefors, (1965)	2.0-4.0 for short delay single row blasting
Kochanowsky, (1964)	1.2-1.5 for field blasting.
Porter, (1971)	1.0-2.0 for production blast
Dick et al., (1973)	1.0 for good fragmentation
Ash and Smith, (1976)	2.0 using delay firing, rows of blastholes perpendicular to dominant jointing
Singh et al., (1981)	2.5-3.0 with instantaneous firing on small scale test
Hagan, (1983)	1.15 is more effective with staggered pattern of holes

Stemming length

The primary function of stemming is to confine the explosive gases so that they have enough time to fracture the rockmass. It has a significant influence on fracture profile and fracture orientation (Daehnke et al., 1997). Improper confinement results not only in wastage of energy and poor fragmentation but also in environmental problems. It was experimentally demonstrated that proper stemming suppresses the airblast and reduces the noise (Linehan and Wiss, 1982).

Insufficient stemming leads to a premature escape of the gases resulting airblast and flyrock. On the contrary, excessive stemming produces boulders coming out from the top part of the bench, poor swelling of the muckpile and an elevated vibration level. Occasionally, if there is a pronounced plane of weakness, particularly a bed or seam of soft material anywhere along the blasthole, stemming may be employed down the hole also. This avoids the release of explosive energy for ground vibration and other undesirable outputs.

Charge length

Grant, (1987) investigated the influence of charge length upon blast vibration. When an explosive charge is initiated,

a detonation front propagates at a characteristic detonation velocity so that the column of charge acts as an external moving source of seismic energy. The problem was studied by using a full scale experiment as well as mathematical modelling, including dynamic finite element techniques. They concluded that as the charge length (L) and thus length to diameter (L/d_e) ratio increases the dominant energy shifts to lower frequencies. When L/d_e equals 1, the dominant frequencies occur near 2 kHz and as L/d_e ratio increases to 88, the dominant frequency decreases to about 0.2 kHz.

Sub-drilling

Sometimes the holes are drilled below the floor level so as to avoid leaving a hump, bootleg, or toe, to obtain a clean breakage at the floor. If the sub-drilling is excessive, the extra explosive energy is utilized in the generation of vibration. Priming below grade level may also increase vibration. Smith, (1979) reported that the blasthole confinement, in terms of burden and spacing dimensions, significantly influences the magnitude of ground vibrations. Optimum sub-drilling depends on geological structural conditions present at the bottom. In thecase of pronounced discontinuity or parting located at grade level, no sub-drilling is required.

Free face

Maximum seismic effects are observed in blasts conducted in a confined medium. As the number of free face increases, the vibration velocity of rocks decreases. In such a case ground vibration may be reduced by as much as 4-5 times compared to blasting in a confined medium (Kuzmenko et al., 1993).

Air decking/decoupling

A study by Wei Wu, (1984) on the effect of blasting-cap scatter on fragmentation and ground vibration showed that a lower level of ground vibration generally occurred for blasts using simulated scatter than from those using simultaneous initiation. It has been suggested by some researchers that in blasting with air decking, the presence of air in air decking plays a critical role in obtaining the advantages (Atchison, 1961; Nicholls, 1962; Smith, 1979; Smith, 1980; Brinkmann, 1982; Warden, 1983). According to Chiappetta, (1987), the air acts as energy accumulator; it stores and further releases energy into the rock medium. Moxon, et al., (1991) suggested that the explosive amount up to 30 % may be reduced without sacrificing production and productivity.

Explosive Parameters

Explosives are substances which, when initiated properly in optimum quantity, releases huge amount of energy with gases in the form of high temperature and pressure along at through the process of called detonation. A liter of modern high explosive will expand to around 1000 liters within milliseconds (ICI, 1997), creating pressures in a blasthole of the order of 10,000 MPa (1,450,000 psi). Temperatures range from 1650-3870˚C and the velocity of detonation (VOD) is so high (2500-8000 m/s) that the power of a single charge is around 25,000 MW. The basic classifications of explosive is illustrated in **Figure 6**.

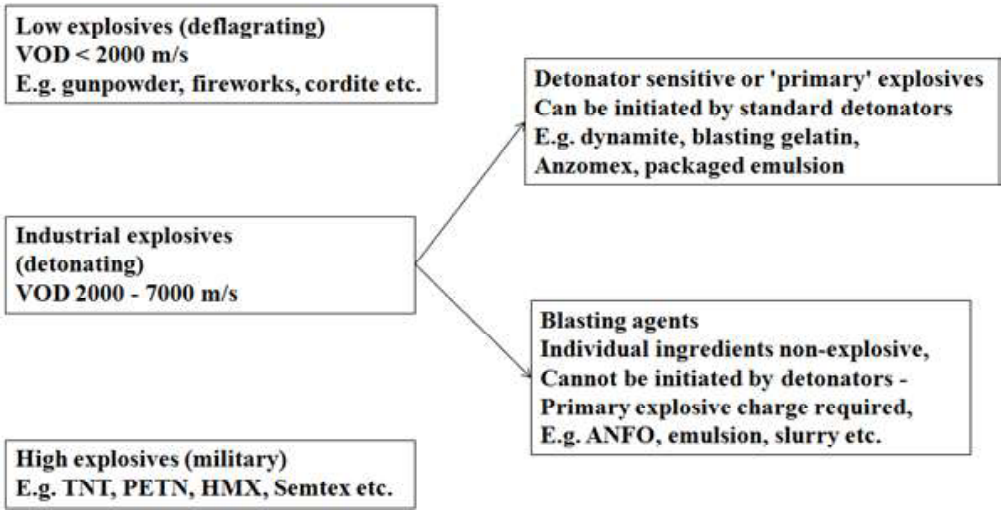


Figure 6. Basic classifications of explosive.

Velocity of detonation (VOD) of explosives

The performance of explosives depends on upon a number of parameters of which VOD is one of the utmost important parameters. The detonation pressure associated with the reaction zone of a detonating explosive is directly proportional to the square of its VOD. It is measured in the C-J plane, behind the detonation front, during propagation through the explosive column. The detonation pressure (P_d) can be estimated by the following formula.

$$P_d = \frac{1}{2}\rho_e (VOD)^2 10^{-6} \qquad \text{.... (1)}$$

Where,
P_d = Detonation pressure (MPa)
r_e = Density of explosive (kg/m^3)
VOD = Velocity of detonation (m/s)

Uniform in-the-hole VOD of explosive is essentially required throughout the blastholes in order to produce sufficient detonation pressure to the blasthole walls. Booster is provided in the explosive column at the bottom to sustain and maintain the VOD for the uniform breakage of rock.

Explosive weight per delay

This is the most important controllable factor influencing ground vibration. Nowadays it is a common practice to use delay blasting. This is practised not only used for reducing vibration but also for obtaining better fragmentation, throw and muck pile profile. Devine et al., (1966) suggested that the peak particle velocity of each component of ground motion can be related to distance and charge weight per delay interval. The effect of distance and charge weight on the vibration level is basic to all quarry blasting studies (Duvall et al., 1963). The ground vibration characteristics depend on a maximum charge per delay in any one of the delay intervals, instead of total charge used in the blast (Nicholls et al., 1971).The function of delay detonation is to separate the pressure front into bundles of energy delivered to rockmass so as to make the events occur in series, independent of breaking.

Total explosive weight in the blasting round and Powder Factor

Most of the researchers consider that the maximum charge per delay plays an important role in the generation of vibration. However, recent field studies revealed that there is a significant effect of the total amount of explosive fired in one round on the magnitude of vibration in the close vicinity of the blasting site (Singh et al., 1994; Siskind, 1994; Singh and Vogt, 1998). Siskind, (1994) further clarified that the complex multi-delayed blasts generates vibration amplitudes up to three times higher compared to those of the same-weight-per-delay single charge.

Venkatesh, (2005) concluded that the total charge in a blast influences insignificantly on the intensity of the ground vibrations for distances between 100 m and 3000 m. With the combination of delay detonators and proper blast design, large scale blasts are feasible without any significant increase in the vibration levels.

Powder factor is interesting and sometimes confusing parameters in respect of the controlling ground vibration. Blasts have been recorded in which the powder factor was reduced 20% from the optimum and vibration levels measured were two or three times higher as a consequence of the confinement and poor spatial distribution of the explosive, causing lack of displacement and swelling energy .

Delay intervals and delay scattering

Delay interval is the difference in firing timings between two groups of holes. But from vibration and fragmentation point of view, the effective delay interval is important. Delay accuracy plays a very important role in the prediction of vibration effects. Not only is the accuracy of the delay important, but the choice of delay is just as important (Wheeler, 1997). The effective delay interval is the difference in arrival times of pulses from two holes fired in successive periods at a given position (Wiss and Linehan, 1978). Kuzmenko et al., (1993) suggested that optimal delay interval with respect to ground vibration varies widely from 10 to 80 ms and more. The delay time should be compatible with the type of rocks, their acoustic properties and period of oscillation of waves. For granites, optimal delay time happens to be 15-35 ms, for limestones, 20-50 ms. In weak rocks, delays with larger intervals (50-80 ms) are preferred. To avoid interference

of seismic waves, the delay interval should exceed 17ms between rows (Kopp and Siskind, 1986). The proper combination of charge weight and timing allows sufficient room for expansion of the rockmass (swelling) between rows in multiple row blasts. Minimum delay interval between rows suggested by different investigators is given in Table 3.3.

Table 3. Minimum delay interval between rows suggested by different investigators.

Investigators	Minimum delay interval between holes
Langefors et al., 1972	5 to 8.3 ms per meter of burden.
Langefors and Kihlstrom, 1978	2 to 5 ms per meter of burden.
Winzer, 1979	3.4 ms/m relief for holes within a row and 7.7 ms/m relief for burden between rows.
Andrews, 1981	3.3 to 17 ms per meter of spacing between adjacent holes in a row. The delay between rows 2 to 3 times that between adjacent holes in a row.
Bergman, 1983	3.3 to 6.6 ms per meter of burden.
Hagan, 1983	8 ms per meter of theburden for soft rock, 4 ms per meter of theburden for hard and massive rock.
Konya and Walter, 1990	3 to 4 ms/m: diabase, porphyrites, gneisses and micaschists, magnetites. 4 to 5 ms/m: Compact limestone and marbles, granites and basalts, quartzites, some gneisses, and gabbros. 5 to 6 ms/s: shales, some limestones, rock salt. 6 to 7 ms: sandstones, coal.
Blair, 1993	Delay interval between blastholes should be 30 ms and a delay interval no greater than 100 ms between rows.

Effect of direction of initiation

Magnitude and frequency of blast induced ground vibrations depend upon the location of monitoring point with respect to the direction of the blast (Singh et al., 1994a). Experiments by Wiss and Linehan, (1978) showed that the seismic and overpressure amplitude measured perpendicular to the direction of the blast were 2 to 2.5 times larger than the measured parallel away from the blast.

Kopp and Siskind, (1986) reported that the orientation of blast and direction of initiation has a noticeable effect on the magnitude of vibration. When the instrument array is located perpendicular to the firing pattern, the seismic and overpressure amplitudes are approximately 2-2.5 times as large as firing away (180°) from the blast. The seismic and overpressure amplitudes for this condition are approximately 3-6 times as large as firing away from instrument array. Singh and Vogt, (1998) found that the PPV measured in the flank opposite to the initiation was almost double the PPV measured in the flank at the initiation end.

Methodology for explosive selection for given rockmass

The acoustic impedance, Z, for any material, is defined as:

$$Z = \rho \times V_p \qquad \text{.... (2)}$$

Where, Z = acoustic impedance,
Ñ = density of material,
V_p= Sonic velocity of material

The rock impedance (Z_1) may be approximated by the product of rock propagation velocity and rock density whereas explosives impedance (Z_2) may be approximated by the product of detonation velocity of explosives and its density. In order to maximise the transfer of explosives energy to the rockmass, the impedance of the explosives should be close to that of the rockmass. When the impedance of the explosives is close to that of the rockmass, explosives energy is better transmitted to the target rock. Under such condition, the maximum pressure transmitted to the rock is nearly equivalent to the

detonation pressure generated inside the pressurised borehole. This is the standard case for the massive rock formation. As the rocks of the mine are highly jointed and/ or fractured the energy utilisation of the explosives with high VOD resulted in to over crushing of the rock fragments. Further, from the observation of impedance value, it is suggested that for getting optimal fragmentation results 80 % of the acoustic impedance value are suitable. When the impedance of the rock is less than the impedance of the explosives, then the major part of the explosives energy transmitted to rockmass will be reflected back as tensile wave and will be responsible for breakage of the rock.

The minimum/limiting condition can be expressed as:

$$Z_1 = Z_2 \quad \text{.... (3)}$$

The Rock Impedance matrix and Explosives Impedance matrix is calculated using the following formulae:

$$[Rock\_impdence] = [Rock_{Sonic\_velocity}] \times [Rock_{Density}] \quad (4)$$

$$[Explosive_{impdence}] = [Explosive_{Inhole_{VOD}}] \times [Explosive_{Density}] \quad (5)$$

UNCONTROLLABLE PARAMETERS OF BLASTING

Geological discontinuities are an integral part of rock mass and significantly influence the blasting operations in mining and civil industries resulting in high excavation cost (Bakar et al., 2013, Roy et al., 2017). The influence of rock mass structure on blasting performance is discussed by many authors (JKRMC 1984, Hagan 1973, Hoek and Bray 1981, Badal 1995; Paswan et al., 2021). The basic mechanisms by which the presence, nature and orientation of rock mass discontinuities affect blasting performance are fairly easy to describe in qualitative terms but very difficult to incorporate in to quantitative design formulae. The structures typically present in a rock mass include joints and faults, bedding planes, foliation and cleavage planes. It is easy to visualise those discontinuities which are favourably oriented with respect to the blast hole will be preferentially extended by a shock wave. The surfaces of pre-existing fractures will act as partial free surface from which unaccountable reflection and refraction of the shock waves will occur. Layered material introduces zones of different impedance and additional boundaries at which complex wave interactions will occur and lead to an increased attenuation of the shock wave. Structures which have been extended or opened by the shock wave will act as a sink for the explosion gases. Interconnected structures may allow the early escape of explosion gases and a rapid reduction of the confined gas pressure ultimately resulting in to poor fragmentation of blasted rock. Many guidelines have been published for the design of blasts to suit particular structures (e.g. Badal, 1995; Lizotte and Scoble, 1994; Hoek and Bray, 1981; Hagan, 1983). The absence of discontinuities makes the blasting of massive rock more predictable, and so more suited to the application of conventional blasting theories. **Figure 7** shows examples of the impact of various geological structures on blasting performance and their remedial measures.

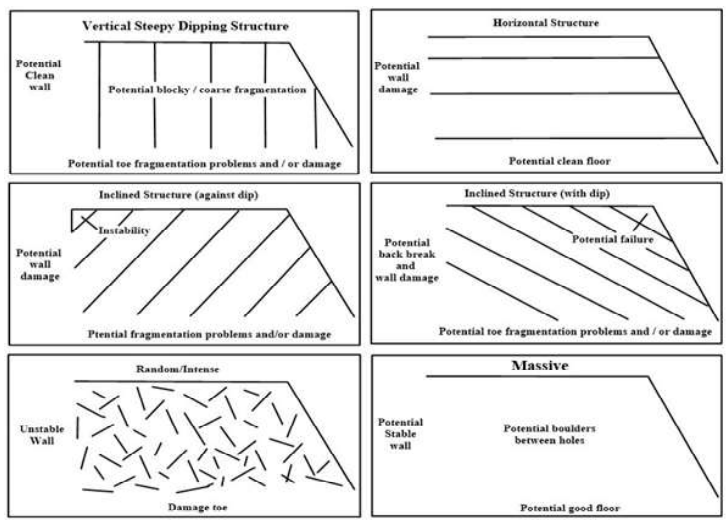


Figure 7. Impact of geological structures on blast performance.

It has been reported by Persson et al. (1994) that blasting in a homogenous isotropic medium naturally does not result in the same fragmentation pattern as when the medium is permeated with discontinuities. If discontinuities are present than the effective area of influence of a hole reduces, because the gaps of the joints will not only hinder the propagation of radial cracks but will also provide easy passage for the gases to escape, thus reducing the borehole pressure. Tariq and Worsey (1996) observed during small scale experiments that 3mm of joint opening reflects back the explosive energy just like a free face, thus no spit plane is produced. The effect of jointing on rock fragmentation documented by Hustrulid (1999) is presented in figure 3.

Relationship between rock geotechnical properties and explosives properties

Rock breakage by explosives involves action of explosives and response of the surrounding rockmass within the realms of energy, time and mass. The rock properties are uncontrollable variables comprising of many

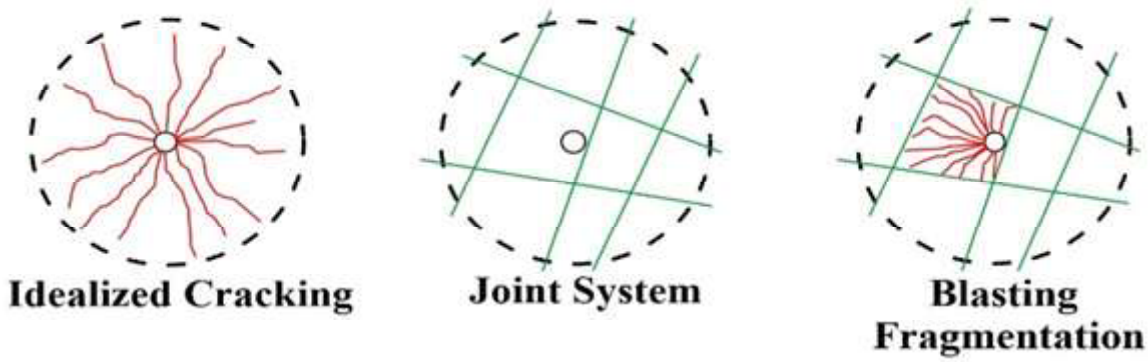


Figure 3. Effect of jointing on fragmentation (after Hustrulid, 1999)

parameters viz. geology, strength, structural discontinuities, the state of weathering of rockmass, etc. It is very difficult to quantify a blast as good or bad on any one single parameter but +-there are a number of parameters which characterises the blast results to be defined a good blast or a bad blast. A good blast comprises of few important criteria which includes optimum fragmentation, optimal muck pile displacement, optimal muck pile profile with ease of digging, lower level of ground vibration generation, lower level of air overpressure generation, non-ejection of flyrock, practically no back or over break and no misfires, etc. In order to achieve the above objectives, selection of suitable explosives is very pertinent. The strength of the rock is an important criterion for selecting the explosives for achieving desired/optimum fragmentation in consonance with the loading equipment in use. In order to meet the optimum fragmentation, the explosive characteristics play a pivotal role with due regard to environmental nuisances i.e. vibration, flyrocks and air over pressure.

The determination of these parameters by direct or laboratory methods is very costly, time consuming and difficult, as the samples tested do not usually include discontinuities and the lithological changes of the rockmass from where they were taken. Rock impedance has a relation with P- wave velocity and density of the rock as well as it can also be calculated using the density of explosive and velocity of detonation. The proportion of energy split between the shock and gas phase will depend on the explosive and rock properties and the degree of confinement. Gama & Jimeno (1993) observed that in intensely structured rock, little breakage is required to obtain a satisfactory size distribution (**Figure 8**).

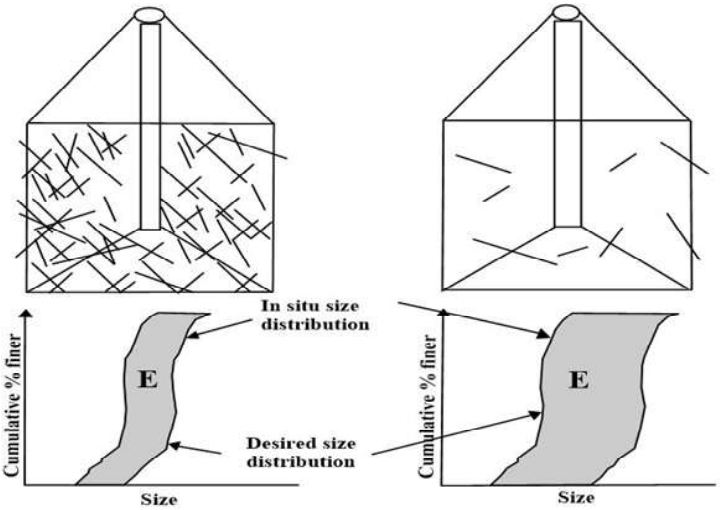


Figure 5. The influence of structure on the energy consumption in blasting to reach the same fragmentation size distribution (after Gama and Jimeno, 1993).

3. Conclusion

Blast designs are abound by rule of thumbs. Burden empirically expressed as a function of hole diameter (i.e., b = 25 to 40D) or that spacing to burden ratios should be between 1 and 2. Following these guidelines, the good blasting results will be obtained. This may in fact be true. However, these sorts of relationships are obtained from sites where a long, and often painful evolution of blast design has yielded good blasting results and this experience has been summarized in terms of relationships between burden and hole diameter and spacing to burden ratio.

The adequacy of blasting practices depends firstly on the objectives of blasting, a presumably good fragmentation is one of these. Good blasting results will be dependent on many factors including rock strength, explosive type,

distribution of explosive within the rock, tie-up and timing and blast pattern dimensions (i.e., burden and spacing) and orientation relative to major joint sets. The source of inconsistent fragmentation probably involves:

- poor and/or inconsistent explosive performance
- poor blast design in the first place
- damaged rock contributing to poor fragmentation
- a change in rock mass conditions
- poor implementation of the blast design in the field

These fragmentation problems can be resolve by proper observations at the site such as some interpretation of the location of boulders (i.e. free face region, within blast volume or near rear of blast), the extent (i.e. length) and spacing of jointing relative to the blast dimensions (burden and spacing), the extent of cracking behind the last row of the blast and therefore the degree of damage induced in the new free face, the extent that joints and bedding planes form the free faces of particles found in the muckpile, the digability of different regions of the blast, any feed-back on size distribution and the crusher or mill performance. Apart from these it is essential to complete the feed-back loop by relating any change in the blasting result to changes in the blast design. This is an essential step as it indicates the direction for future improvement.

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Air Pollution Tolerance of Common Plant Species for Greenbelt Development Around the Coal Mining Areas of Jharia Coalfield, India

Shilpi Mondal* Gurdeep Singh*

INTRODUCTION

In recent years air quality in metropolitan cities of developed nations has degraded significantly due to massive rise in vehicular transportation, fossil fuel combustion in industries, coal mining activities and the loss of vegetation cover (Srivastava et al., 2013; Pandey et al., 2014; Amegah and Agyei-Mensah, 2017; Jena et al., 2019; Mondal et al., 2020). Vehicular emissions and mining activities are regarded as significant sources of air pollution such as particulate matter, CO, SOx, NOx, trace elements, and polycyclic aromatic hydrocarbons (PAHs) (Jena and Singh, 2016; Mondal and Singh, 2021). numerous experts have investigated the negative impacts of air pollution on plants all over the world (Sadeghian and Mortazaienezhad; 2012; Govindaraju et al., 2012; Pandey and Agarwal, 2014; Zhang et al., 2016). As Green belts are efficient means to mitigate air emissions by trapping particulate matter and capturing gaseous contaminants, environmental activists and policymakers therefore stressed the importance of a permanent green belt in and near urban areas, and along highways, to decrease the consequences of poisonous fine particulates (Sharma et al., 2017; Karmakar and Padhy, 2019; Alotaibi et al., 2020) and to use as bioindicator of air pollution (Gupta et al., 2011). Plants can eradicate air pollution by three mechanisms: leaf absorption, particulate accumulation on leaf surfaces, and particulate deposition on the leeward side of the leaf due to the wide surface area of the leaves, which acts as a sink (Prajapati and Tripathy, 2008; Roy et al., 2020; Javanmard et al., 2020). After penetration, the pollutants block the stomata and causes a decrease in the foliar pH due to the prevalence of sulphate (SO_4^{2-}) and nitrate (NO^{3-}) ions content in the dust fall (Gupta et al., 2016). Leaves of plants serve as an environmental sink as it provides wide surface area for trapping and accumulation of air pollutants (Rai, 2016). Leaves are one of the most effective pollutants trapping systems through absorption and diffusion of particulates and gaseous pollutants (HF, SO_2, NOx and other trace elements) (Balasubramanian et al., 2018). However, air pollutants, on the other hand, may have a negative impact on plant growth by affecting the biochemical parameters, photosynthetic activity, morphological characteristics, and seed germination (Kaur and Nagpal, 2017). Many plants can adapt to the changing environments through adaptations in biochemical parameters, especially in chlorophyll, ascorbic acid, leaf pH, and relative water content. The changes in the above biochemical parameters can be taken into account in estimating the Air Pollution Tolerance Index (APTI) of plant species while classifying the plant species as sensitive, intermediate, or tolerant to air pollutants (Rai and Panda 2014; Molnár et al. 2020).

The Air Pollution Tolerance Index (APTI) is regarded as an effective method for identifying bioindicator plants, which is calculated by four biochemical parameters such as Ascorbic acid (AA), Total chlorophyll content (TCH), Leaf extract pH (pH), and Relative water content (RWC) (Pandey et al. 2015; Kaur and Nagpal 2017). AA serves as a coenzyme in the metabolism of carbohydrates, fats, and proteins, as well as the production of nucleic acid in RNA, in photosynthetic activity and growth of plants (Mazher et al., 2011). TCH, as one of the most important components of energy generation in green plants, clearly impacts plant health, which is greatly influenced by several environmental factors (Agbaire and Akporhonor, 2014). RWC is a key determinant of plant protoplasmic permeability. As a result, plants with greater RWC levels could be more tolerant to air pollutants (Nayak et al., 2014). The pH of plants is strongly related to air pollution, particularly sulphur dioxide. Plants having lower pH seem to be more sensitive and with pH approaching 7 are more tolerant (Swami and Chauhan, 2015).

Plants with higher APTI values are regarded as the most tolerant of polluted environments and are ideal for the establishment of green belts, while plants with lower APTI values are the most sensitive and serve as bioindicator species of air pollution (Roy et al., 2020).

**Department of Environmental Science and Engineering, Indian Institute of Technology (Indian School of Mines), Dhanbad -826004, Jharkhand, India*
Corresponding Author: shilpi.iit.ism@gmail.com[1], gurdeep@iitism.ac.in

MATERIALS AND METHODS

Study Area

The present study was carried out at JCF, one of India's largest coalfields, located in the heart of the Damodar River Valley, which encompasses an area of 450 square meters (Fig. 1). Drilling, blasting, associated mining activities, mine fire, vehicular load at the coal mines contribute significantly to a higher concentration of air pollutants around JCF. A total of nine locations were selected around the coal mining areas of JCF for sampling of leaves viz. Tetulmari PS (S1), Loyabad PS (S2), Lodhna PS (S3), Kenduadih PS (S4), Bastacolla Colliery (S5), Sijua Stadium (S6), Jogta 14 pit (S7), Katras (S8), and Kujama Colliery (S9). A reference site, IIT(ISM) Dhanbad (S10) was selected for comparing the biochemical and physiological changes that occur in vegetation community structure.

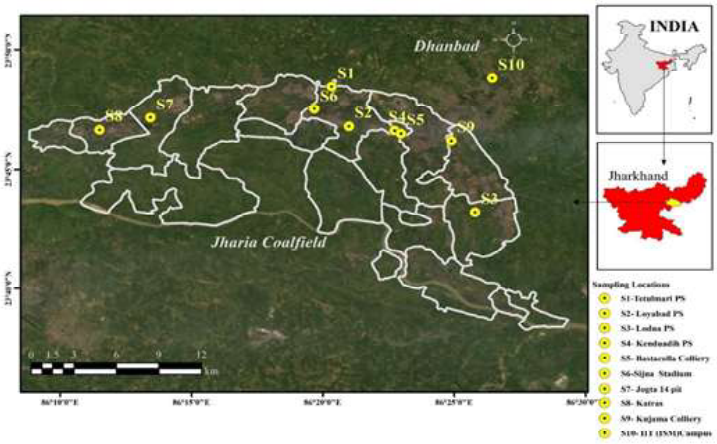

Fig.1. Map of the Study Area

Selection of plant species

Leaf samples of ten plant species, namely, *Ficus religiosa* L. (P1), *Mangifera indica* L. (P2), *Alstonia scholaris* (L.) R. Br. (P3), *Azadirachta indica* A. Juss. (P4), *Moringa oleifera* Lam. (P5), *Ficus bengalensis* L. (P6), *Bougainvillea spectabilis* Willd. (P7), *Rosa indica* L. (P8), *Calotropis gigantea* (L.) W.T. Aiton (P9), *and Catharanthus roseus* (L.) G. Don. (P10) were collected during January 2019 to January 2020 by ensuring rainfall and temperature conditions. These fifteen species are found at both the study area i.e. the coal mining areas of JCF and reference site, IIT(ISM) Dhanbad.

Estimation of pH of leaf extract

0.25gms of leaf samples were homogenized in 10ml double distilled water, centrifuged at 5000 rpm for 5 mins, and the pH content of the leaf extract is estimated from the supernatant using pH meter (Karmakar and Padhy, 2019).

Estimation of Total Chlorophyll content (TCH) of leaf extract

The total chlorophyll content of the leaf extract was estimated using the method introduced by Arnon, 1949. 0.25gms of leaf samples were crushed and homogenized with 10ml of 80% acetone, centrifuged at 5000 rpm for 5 mins. The supernatant was volume makeup up to 25ml was done with 80% acetone. The optical density of the leaf extract was calculated at 645nm and 663 nm using a UV-Visible spectrophotometer. Total chlorophyll content (mg/g) of the leaves was calculated according to the following formulae:

Chlorophyll a (mg/g) = $12.7 \times A_{663} - 2.69 \times A_{645}$

Chlorophyll b (mg/g) = $22.9 \times A_{645} - 4.68 \times A_{663}$

Total Chlorophyll Content (mg/g) = $20.2 \times A_{645} + 8.02 \times A_{663}$

Estimation of Relative Water Content (RCH) of leaves

RCH of leaf samples was calculated by the given formula, explained by Henson et al., 1981.

$$RCH = \frac{fresh\ weight - dry\ weight}{turgid\ weight - dry\ weight} \times 100$$

Fresh leaves after taking to the laboratory were washed properly with distilled water. After removing excess water, fresh weights of the leaves are taken. After that, turgid weights of the leaves were taken by immersing them in distilled water overnight. Then the leaves were blotted and dried at 115 C for 2 hrs. to calculate the dry weight

Estimation of Ascorbic Acid content (AA) of leaf extract

Ascorbic acid content of the leaf samples was estimated using the method explained by Keller and Schwager (1977). Fresh leaves were homogenized with a mixture of oxalic acid and EDTA, homogenized and the supernatant was collected. Titration was then done with 2, 6-dichlorophenol indophenol until the solution turns pink. The optical density of the solution was measured at 520 nm before (Es) and after adding one drop of ascorbic acid to the solution (Et). At the same time, the optical density of 2, 6-dichlorophenol indophenol was also measured at the same wavelength (Eo). The

concentration of Ascorbic acid was calculated by the given formula after preparing a standard curve with the same method.

$$AA\ (mg/g) = [Eo - (Es - Et)] \times \frac{V}{W} \times 1000$$

Where W = weight of the fresh leaves, V = total volume of the mixture, and the value of [Eo – (Es – Et)] was evaluated by the standard curve.

Calculation of Air Pollution Tolerance Index (APTI)

APTI was calculated by the following equation, as explained by Singh and Rao, 1983.

$$APTI = \frac{A(T+P)+R}{10}$$

Where, A = Ascorbic acid content of the leaves, T = Total chlorophyll content, P = pH of leaf extract and R = relative water content.

RESULTS AND DISCUSSION

pH

Variability in pH values was noted among species (Table 1), which signifies the differences in responses among each plant species for a particular pollutant in a polluted environment (Kaur and Nagpal 2017). Comparison in pH values was estimated between the coal mining areas of JCF and the Reference site. The values of pH at Reference sites were significantly lower than the sites at coal mining areas of JCF for all the plant species. pH values of all the plant species at JCF ranged between 5.00 - 9.00, with the lowest value was observed for *C. roseus* (5.07 ± 0.43), whereas the highest value was observed for *F. religiosa* (8.68 ± 0.43). In the study area, higher pH values of *F. religiosa* (8.68), *F. bengalensis* (8.60), *A. scholaris* (7.85), *M. oleifera* (6.60) and *A. indica* (6.50), indicated more tolerance towards air pollution even in a polluted atmosphere. However, plants with lower pH values are adversely affected by air pollution due to their higher stomatal sensitivity.

Relative Water Content (%)

The results obtained revealed that the values of RWC (as a percentage) varied across 10 different plant species (Table 1), the highest being observed at the Reference site when compared to the sites of JCF. Among all the species, RWC was found to be highest for *F. bengalensis* and lowest for *C. roseus* at sites of JCF and Reference respectively. Apart from these, RWC of plants like *F. religiosa, M. indica, A. scholaris,* and *M. oleifera* were found to be above 90% in the study area. Leaf water content is responsible for growth, transpiration, respiration, and maintenance of physiological balance (Kaur and Nagpal, 2017), as a result higher RWC enhance the resistivity of the plants during drought (Geravandia et al. 2011). As RWC is directly related to protoplasmic permeability, a higher proportion of RWC in plants improves resistance and enables the plant more tolerant of stress caused by polluted air (Jyothi and Jaya 2010; Gupta et al., 2016).

Total Chlorophyll content

The variation in chlorophyll content of leaves of 15 different plant species at the sites of JCF and the Reference is depicted in Table 2. The chlorophyll content of leaves observed in the plants of Reference site was found to be much higher than the sites of JCF for all the studied plant species. The findings of this study were consistent with the findings of a recent study done by Alotaibi et al., 2020, who reported that the TCH contents of five plant species (*A. lebbeck, E. camaldulensis, F. altissima, P. juliflora, and Z. spinachristi*) decreased as the concentration of pollution increased. Gaseous pollutants (SO_2, NO_2, and particulate matter), high temperature, drought, particulate matter deposition, and soil pollution can all lower chlorophyll concentration in plants (Karmakar et al., 2016).

Ascorbic Acid content

Significant variation in ascorbic acid content was observed among the studied plant species at JCF and Reference sites (Table 2). Higher ascorbic acid content was observed at the sites of JCF when compared with Reference site. From the results, it is observed that *F. bengalensis* showed the highest Ascorbic acid content (14.11 ± 0.32), followed by *F. religiosa* (12.32 ± 0.54), *L. indica* (9.56 ± 0.42), *A. scholaris* (9.34 ± 0.28), and *M. indica* (9.08 ± 0.23), whereas lowest was found for *C. roseus* (5.89 ± 0.52) and *C. gigantea* (6.09 ± 0.32). The higher ascorbic acid content recorded in leaves of polluted locations showed the tolerance of plant species to their developing environment's heightened air pollution levels (Zhang et al., 2016; Yadav and Pandey, 2020).

Air Pollution Tolerance Index (APTI)

The four biochemical parameters viz. Total Chlorophyll, pH, Ascorbic acid and Relative water content were used

to calculate the APTI of an individual plant species (Table 1). APTI investigates the vulnerability of particular plant species to air pollution (Singh et al., 1991). Higher values of APTI of the plant species were found at sites of JCF, whereas lower APTI values were found at the Reference site. This finding is confirmed by equivalent research published in the literature (Karmakar and Padhy, 2020). The potential of a plant species to withstand air pollution may be assessed using the APTI index. Plants having a higher APTI index value are regarded tolerant to air pollution and may be utilized to reduce pollution levels. The species with a lower index (sensitive species) on the other hand, might be utilized as an indicator of air pollution (Singh and Rao, 1983). According to the values of APTI, in the present study, *F. bengalensis* (25.21 ± 0.95) was found to be the most tolerant, followed by *F. religiosa* (23.02 ± 0.21). Contrastingly, *C. roseus* (11.63 ± 0.87) and *C. gigantea* (11.87 ± 0.34) were found to be sensitive to air pollution due to their lower APTI scores.

, Ascorbic acid) with APTI of studied plant species at the sampling locations of JCF as well as Reference site

Plant Species	Sites	Relative Water Content (%)	pH	Total Chlorophyll (mg/g)	Ascorbic Acid (mg/g)	APTI
P1	JCF	92 ± 2.45	8.68 ± 0.43	2.54 ± 0.31	12.32 ± 0.54	23.02 ± 0.21
	R	95 ± 3.06	7.09 ± 0.29	2.87 ± 0.26	5.86 ± 0.62	15.34 ± 0.37
P2	JCF	90 ± 3.56	6.35 ± 0.54	2.32 ± 0.24	9.08 ± 0.23	16.88 ± 0.65
	R	92 ± 1.36	5.34 ± 0.28	2.65 ± 0.37	4.45 ± 0.33	12.76 ± 0.26
P3	JCF	89 ± 3.23	7.85 ± 0.42	2.42 ± 0.15	9.34 ± 0.28	18.50 ± 0.43
	R	90 ± 2.67	5.87 ± 0.43	2.77 ± 0.11	5.26 ± 0.17	13.55 ± 0.32
P4	JCF	85 ± 3.12	6.50 ± 0.43	2.35 ± 0.27	8.32 ± 0.43	15.87 ± 0.21
	R	89 ± 2.85	5.06 ± 0.43	2.50 ± 0.36	4.21 ± 0.14	12.09 ± 0.87
P5	JCF	86 ± 2.22	6.6 ± 0.43	2.41 ± 0.86	8.56 ± 0.56	16.32 ± 0.66
	R	88 ± 1.87	5.54 ± 0.43	2.66 ± 0.72	4.87 ± 0.15	12.80 ± 0.54
P6	JCF	93 ± 2.14	8.60 ± 0.43	2.67 ± 0.42	14.11 ± 0.32	25.21 ± 0.95
	R	96 ± 2.37	7.00 ± 0.43	2.94 ± 0.36	6.08 ± 0.11	15.65 ± 0.43
P7	JCF	87 ± 3.44	7.13 ± 0.43	2.43 ± 0.87	8.78 ± 0.65	17.10 ± 0.73
	R	90 ± 2.86	6.87 ± 0.43	2.61 ± 0.22	4.76 ± 0.39	13.52 ± 0.71
P8	JCF	78 ± 3.76	5.23 ± 0.43	1.28 ± 0.43	7.52 ± 0.54	12.70 ± 0.21
	R	80 ± 2.90	5.07 ± 0.43	1.46 ± 0.33	3.32 ± 0.12	10.17 ± 0.54
P9	JCF	77 ± 2.43	5.65 ± 0.43	1.19 ± 0.31	6.09 ± 0.32	11.87 ± 0.34
	R	81 ± 2.45	5.43 ± 0.43	1.33 ± 0.30	2.67 ± 0.16	9.91 ± 0.49
P10	JCF	80 ± 2.66	5.07 ± 0.43	1.08 ± 0.21	5.89 ± 0.52	11.63 ± 0.87
	R	82 ± 2.70	5.02 ± 0.43	1.25 ± 0.11	2.14 ± 0.22	9.17 ± 0.32

CONCLUSIONS

As this study demonstrates, individual tree species react differently to air pollution. According to our findings, polluted metropolitan areas led to a decrease in leaf area, total chlorophyll content, and leaf relative water content. As suggested by the data, suitable tree species should be selected depending on their tolerance to pollution load and utilizing Air Pollution Tolerance Index (APTI). Plant with lower APTI (*C. roseus and C. gigantea*) act as sensitive species can be used as a bioindicator, whereas plant species exhibiting higher APTI values (*F. bengalensis, F. religiosa, M. indica, A. scholaris, A. indica and M. oleifera*) are tolerant to air pollution, that can be proposed for planting programs in polluted areas to reduce pollution levels. Significantly, the tolerant plant species can be implemented into a greenbelt design of Jharia Coalfield, to aid in the long-term control of air pollution effectively.

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Assessing Utilization of Underground Minewater of Jharia Coalfield for Drinking Purposes Using an Integrated Modelling Approach

Pritam Mazinder Baruah* Gurdeep Singh*

INTRODUCTION

The demand for water resources is set to increase over the years with the increasing population coupled with a rise in industrial growth, urbanization, and agricultural activities. Large parts of India have already been declared as water stressed and face a critical situation which has been attributed to mismanagement of water resources. Accessing safe potable water is a problem that continues to persist in many areas. Skewed water availability between different regions and between different people in the same region compounded by intermittent and unreliable water supply system risk causing social unrest. The optimal utilization and management of available water resources thus assumes significance. Minewater is discharged in large quantities from the coal mines and only a small percentage of it is used in reclamation and dust suppression while the remainder of the minewater discharges find its way into natural drainage systems (Singh et al. 2012). Minewater discharged during mining operations is a valuable water resource, but it runs the risk of getting contaminated from domestic and industrial effluents, causing adverse effects on human health, and stymying the growth and development of the region (Neogi et al. 2018; Adimalla and Qian 2019). Water resources management is vital for sustainability in densely populated and industrialized regions. This becomes more relevant in water-stressed regions, prone to contamination with limited access to potable water (Badham et al. 2019; Xiang et al. 2021). Efficient water resource management requires a significant capital investment in smart infrastructure, efficient treatment technologies, quantifiable indicators, and monitoring tools (Behmel et al. 2016; Platikanov et al. 2019). The installation of specific infrastructures has been proposed and implemented to ameliorate potable water quality and to ensure its future availability. In this context, an integrated approach to assess the water quality of a region by using IWQI and HPI model, reduces time and effort expended in the treatment processes. Based on the values of the integrated indices, corroborated by chemometric methods like Principal Component Analysis (PCA) and Hierarchical Cluster Analysis (HCA), appropriate treatment technologies can be set up, thus, adopting a comprehensive water resource management strategy. While traditional water quality indices (WQI) consider the values of physicochemical parameters below the acceptable limits only to be good without accounting for the ion-deficiency in the water, giving an erroneous assessment of water quality, IWQI uses both- acceptable/ desirable limits and permissible limits in its assessment of water quality from the aspect of ion concentration in the minewater.

In this paper, we seek to give an insight on the assessment of potability of minewater pumped out from Jharia Coalfield, using an integrated approach which constitutes application of the IWQI Model, HPI Model and Multivariate Statistics to fill the research gap of comprehensive evaluation of water quality of the study area by overcoming the limitations of traditional WQI model by incorporating both the drinking water threshold levels in its calculation as well as accounting for the presence of heavy/trace metals present in the sampled water, giving a holistically accurate assessment of the minewater quality.

MATERIALS AND METHODS

Study site

Jharia Coalfield is a sickle-shaped terrain that spreads across a 450 km^2 area between 23°37' N – 23°52' N latitudes and 86° 09'E–86° 30' E longitudes with an altitude averaging 220 metres above mean sea level, situated in Dhanbad district, in the Indian state of Jharkhand. It is located at the centre of the Damodar valley, bounded by the Eastern Railway in the north and the Damodar river in the south (Singh et al. 2018). The sampling sites selected for the current study comprised of 15 operational minewater treatment plants (MWTPs) spread across Jharia Coalfield (Fig.1). The MWTPs are situated in the mining areas of Mudidih, West Mudidih, Ramkanali, Kharkharee, Sinidih, Muraidih, Shatabdi, Bastacolla, Victoria, Bera, Kuya, South Tisra, Khas Kusunda, Pootkee-Balihari (P.B.) Area, which are owned by Bharat

**Department of Environmental Science and Engineering, Indian Institute of Technology (ISM), Dhanbad – 826004, Jharkhand, India*
Corresponding Author: pritam.iit.ism@gmail.com, gurdeepsingh.iit.ism@gmail.com

Coking Coal Limited (BCCL). The groundwater level distribution in the study area for the pre-Monsoon and post-Monsoon seasons of the hydrological year 2019-2020 (Fig. 1A, 1B) was prepared by interpolating groundwater data of the region using IDW technique in QGIS Long-Term Release (LTR) 2.8.1 software.

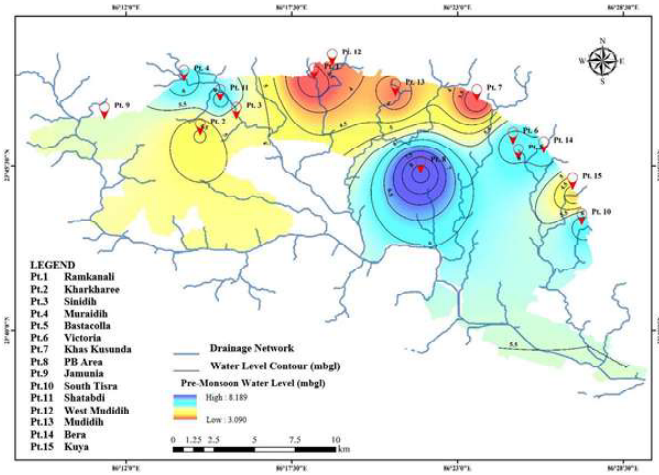

Figure 1 A: Map of study area depicting groundwater level distribution during pre-Monsoon season

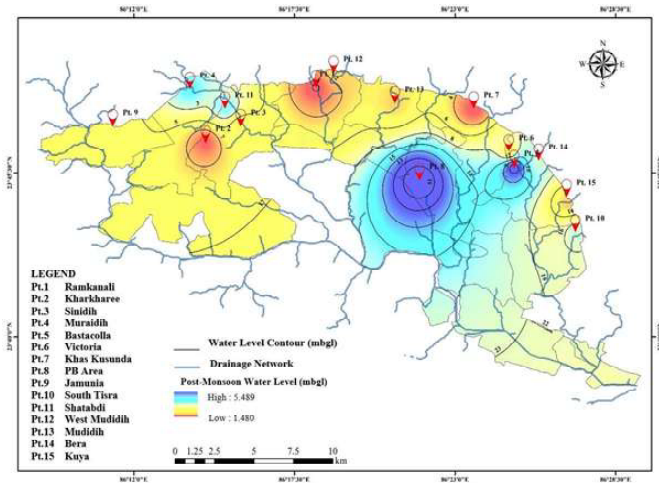

Figure 1 B.: Map of study area depicting groundwater level distribution during post-Monsoon season

Sampling and analysis

Two sampling campaigns were carried out in the pre-monsoon and post-monsoon seasons of March-April and October-November, respectively for the hydrological year 2019-2020. Ninety underground coal minewater samples (3 composite samples from each of the 15 minewater treatment plants operating in JCF, Jharkhand) were collected from the storage tank outlets of the minewater treatment facilities in high-density polyethylene bottles (pre-washed) of 1 litre capacity for the year 2019-2020. In each site, three field duplicates were collected and well mixed in situ subsequently. A total of 14 physicochemical parameters and 9 trace/heavy metals (Iron, Manganese, Zinc, Nickel, Lead, Copper, Chromium, Arsenic and Cadmium) determining water quality of the region were analysed. Type-I (18.2 M&!cm) Milli-Q water was used to prepare all the solutions. EC, pH, and TDS values were gauged on-site using Multiparameter pH Tester 35 Ecotech instrument. Total alkalinity (titration with 0.02N H_2SO_4 using phenolphthalein and methyl orange indicator), total hardness (titration with EDTA), sulphate (turbidimetric method), nitrate (spectrophotometric method), chloride (argentometric method) and fluoride (SPADNS method) were analysed using UV-visible spectrophotometer (UV-1800 Shimadzu), as per the methods prescribed in APHA 2017. Duplicate blanks and a laboratory water standard were analysed for quality control. EDTA titration method was used to analyse calcium while magnesium was determined by multiplying the magnesium hardness by a factor of 0.2431. Potassium and sodium were determined by the flame photometer (ESICO, M-1385). The trace/heavy metal concentration in the minewater was determined after digestion of the samples with 1 N HNO_3 prior to its analysis, using GBC Avanta PM Atomic Absorption Spectrophotometer in flame mode at the recommended wavelengths. The spatial distribution of the integrated water quality index was drawn using IDW technique in ArcGIS 10.2.2 software.

Calculation of IWQI

IWQI was applied to the coal minewater samples to evaluate the minewater quality and its suitableness for consumption. Studies on potability of water have revealed its beneficial effects on human health to occur at acceptable concentrations (Li and Wu 2019). Therefore, acceptable limits (AL) and permissible limits (PL) values of the water quality parameters are assigned by the BIS based on the hazard it poses to human health (Table 1). The values between the two threshold limits fall under the range which can also be represented as the difference in deficit (15%) of the range of the specific water quality parameter from its permissible limit (PL) to give Modified Permissible Limit (MPL) given in Table 1. The percentage (%) deficit can be changed as per the situation and is used to buy time to alleviate the existing contamination levels, preventing contamination from crossing the threshold level which otherwise could adversely affect the environment around it (S. Mukate et al. 2019). The values

that fall within the AL and MPL are considered ideal for consumption. When the value of xth water quality parameter (Q_x) is higher than AL but lower than MPL i.e., AL $\leq Q_x \geq$ MPL, the value of the sub index (SI_1) is taken as zero. When the value of xth water quality parameter is lesser than the acceptable limit i.e.,

$$Q_x \leq \text{AL, then } SI_2 = \frac{(AL - Q_X)}{AL}$$

When the water quality parameter Q_x has a higher value than the MPL i.e., $Q_x \geq$ MPL,
then

$$SI_3 = \frac{(Q_x - MPL)}{MPL}$$

To compute the sub-indices, division of the difference in the specific water quality parameter (Q_x) with its acceptable limit (AL) or Modified Permissible Limit (MPL) is done by its respective AL or MPL. This is done to obtain a homogenized value for identification of increased or decreased concentration of a specific parameter with respect to its AL or MPL. These computed values are added to get the Integrated Water Quality Index (IWQI).

$$IWQI_X = \sum_{y=1}^{n} SI_{xy}$$

where SI_{xy} = value of sub-index of xth sample and y^{th} water quality parameter.
The above equation is used in the calculation of IWQI. The quality of the analysed minewater samples were classified as- excellent (< 1), good (1–2), poor (2–3), very poor (3–5) and unsuitable (> 5) depending upon their IWQI value (Table 2) which depends upon the ionic (Mg^{2+}, Ca^{2+}, K^+, Na^+, Cl^-, F^-, SO_4^{2-}, NO_3^-) concentration along with pH, TDS, total hardness and total alkalinity present in the samples.

Table 1 Water quality standards (BIS 2012) with modified permissible limits for drinking

Parameters	Acceptable Limit (AL)	Permissible Limit (PL)	Range	Modified Permissible Limit (MPL= PL- 15% of Range)
pH	6.5	8.5	2	8.2
Total Hardness (TH)	200	600	400	540
Total Alkalinity (TA)	200	600	400	540
Total Dissolved Solids (TDS)	500	2000	1500	1775
Calcium (as Ca^{++})	75	200	125	181.25
Magnesium (as Mg^{++})	30	100	70	89.5
Sodium (as Na^+)	-	-	-	-
Potassium (as K^+)	-	-	-	-
Chloride (as Cl^-)	250	1000	750	887.5
Fluoride (as F^-)	1.0	1.5	0.5	1.425
Bicarbonate (HCO_3^-)	-	-	-	-
Sulphate (SO_4^{2-})	200	400	200	370
Nitrate (NO_3^-)	45	45	0	45

† No guideline values have been established for some of the parameters as its concentration levels do not concern health. All the parameters are in mg L^{-1}, barring pH.

Table 2 Categorization of quality of minewater based on IWQI (S. Mukate et al., 2019)

IWQI values	Category	Assessment of water quality
< 1	Excellent	Ideal for potable purposes
1-2	Good	Fit for potability and domestic purposes
2-3	Poor	Only for domestic purposes
3-5	Very Poor	Need attention prior to its usage
> 5	Unsuitable	Harmful to human health

Calculation of HPI

The trace/heavy metal contamination affecting minewater quality were evaluated by computing the analysed samples with HPI, an indexing model which incorporates weighted arithmetic quality in assigning weights (w_x) ranging from 0 to 1 to individual metals (Singh and Kamal 2017). The value for critical pollution index is set at 100 for drinking water (Bhardwaj et al. 2017). The standard for the analysed minewater quality parameter is designated as S_x which is in inverse proportion to the unit weightage (W_x) of the respective minewater quality parameter.
The formula proposed by Mohan et al. (1996) was utilized in calculation of HPI

$$HPI = \frac{\sum_{x=1}^{n} W_x Q_x}{\sum_{x=1}^{n} W_x}$$

where W_x denotes unit weightage of xth minewater quality parameter, sub-index of the xth parameter is denoted by Q_x and n is the number of minewater quality parameters analysed. is computed by applying the formula:

$$W_x = \frac{p}{S_x}$$

where the permissible limit (BIS 2012) of xth minewater quality parameter is denoted by S_x and p is assigned to be the proportionality constant. Q_x denotes sub-index of xth minewater quality parameter and was calculated using the formula:

$$Q_x = 100 \times \frac{M_x}{S_x}$$

where S_x is the permissible limit (BIS 2012) for the xth parameter and M_x which is expressed in ìg L^{-1} denotes monitored value of xth minewater quality parameter. In this study, HPI values were grouped into three categories which have been demarcated as high (>30), medium (15-30) and low (<15) depending upon trace/heavy metal concentration in the analysed minewater (Giri and Singh 2014; Panigrahy et al. 2015; Tiwari et al. 2015).

RESULTS AND DISCUSSION

A descriptive statistic of the minewater quality parameters of the minewater samples collected from the study area across two campaigns (pre-Monsoon and post-Monsoon) for the year hydrological year 2019-2020, with its respective acceptable limit (AL) and permissible limit (PL) is represented in Table 3. A detailed description of the hydrological compositions of the major cations, anions, and the charge balance errors (%) of the study area for the hydrological year 2019-2020 is given in Supplementary Table 2.

The cationic concentration of major ions in the coal minewater of JCF in order of their relative abundance was Calcium > Magnesium > Sodium > Potassium, while the anionic concentrations were in the order: Sulphate > Bicarbonate > Chloride > Nitrate > Fluoride. The value for the total cationic charge TZ^{+} ranged between 3.95 and 11.9 meq L^{-1}, (mean- 8.85 meq L^{-1}). The total anionic charge (TZ^{-}) ranged between 4.25 meq L^{-1} and 12.83 meq L^{-1}, having an arithmetic mean of 9.38 meq L^{-1}. Charge balance error upto N±5% was considered acceptable for all the minewater samples (N). 37% of the samples showed a charge balance error exceeding 5%, exhibiting a deficit in total cations due to an anionic excess because of accumulation of contaminated load due to mining operations near the area of study.

Considering the most important trace/heavy metals from previous investigations in JCF, whose concentration levels affect human lives, nine trace/ heavy metals were chosen for this study. Their order of relative abundance was found to be Fe > Mn > Zn > Ni > Pb > Cu > Cr > As > Cd. The trace/heavy metal concentrations in most of the analysed samples, barring a few exceptions, were lower than the permissible limits set by BIS 2012. Fe, Mn, Pb, Cu and Ni were observed to occur in concentrations exceeding the acceptable limits at some sites. Fe concentration ranged from 226 to 626 µg L^{-1} (mean- 421 µg L^{-1}) across the study sites. 74% of the samples had Fe concentration exceeding the maximum acceptable limit of Fe (300 µg L^{-1}). The Mn concentration ranged from 4.5 µg L^{-1} to 697.4 µg L^{-1} (mean- 118.4 µg L^{-1}). 27% of the analysed samples had Mn concentration exceeding the permissible limit of 300 µg L^{-1}, with Bera (697.4 µg L^{-1}) and Kuya (594 µg L^{-1}) having the highest Mn concentration. The concentrations of Pb varied between 7.6 µg L^{-1} and 24.8 µg L^{-1} (mean- 12.3 µg L^{-1}), with 60% of the analysed samples crossing the maximum acceptable limit of 10 µg L^{-1}. Cr, Cd and As, which are highly toxic trace/ heavy metals were within the permissible limits set by BIS 2012. Ni concentration exceeded the maximum acceptable limit of 20 µg L^{-1} in 27% of the analysed minewater samples.

Table 3: Statistical summary of physico-chemical characteristics of the sampled minewater

Parameters	Unit	Coal minewater samples				BIS (2012)		WHO (2011)
		Min	Max	Mean	SD	AL	PL	
Major ions								
pH	-	6.5	8.3	7.7	0.5	6.5	8.5	6.5-8.5
EC	µS cm^{-1}	568	1389	1016	212.4	-	-	1500
TDS	mg L^{-1}	341	953	674	152.9	500	2000	600-1000
TH	mg L^{-1}	149	719	444	151.8	200	600	-
TA	mg L^{-1}	23	403	179	107.3	200	600	-
Ca^{++}	mg L^{-1}	34	131	82	29.59	75	200	75
Mg^{++}	mg L^{-1}	16	116	58	28.00	30	100	125
Na^{+}	mg L^{-1}	9.7	133	34.5	26.13	-	-	200
K^{+}	mg L^{-1}	1.2	12.4	5.0	2.88	-	-	12
TZ^{+}	meq L^{-1}	3.95	11.9	8.85	2.04	-	-	-
Cl^{-}	mg L^{-1}	9.0	151.4	42.3	30.57	250	1000	250
F^{-}	mg L^{-1}	0.09	2.58	0.67	0.49	1	1.5	1.5
HCO_3^{-}	mg L^{-1}	29	492	220	129.3	-	-	125-130
SO_4^{2-}	mg L^{-1}	48	629	293	160.9	200	400	250
NO_3^{-}	mg L^{-1}	0.25	34.59	7.40	9.15	45	45	50
TZ^{-}	meq L^{-1}	4.25	12.83	9.38	2.09	-	-	-
Trace/heavy metals								
Fe	µg L^{-1}	175	626	374	122.4	300	300	300
Mn	µg L^{-1}	3.1	697.4	100.5	187.2	100	300	100-400
Cd	µg L^{-1}	0.01	0.37	0.10	0.08	3	3	3
Cu	µg L^{-1}	3.7	60.9	14.3	12.80	50	1500	2000
Ni	µg L^{-1}	5.7	32.7	14.7	7.31	20	20	70
Pb	µg L^{-1}	7.2	24.8	11.9	4.64	10	10	10
Zn	µg L^{-1}	13	109	46	28.61	5000	15000	3000
Cr	µg L^{-1}	2.9	24.0	6.9	5.23	50	50	50
As	µg L^{-1}	0.2	25.2	5.4	7.40	10	50	10

† No guideline values have been established for some of the parameters as its concentration levels do not concern health. SD = Standard Deviation.

Physico-chemical characteristics of minewater

The analysis of physicochemical parameters of the pumped-out coal minewater samples from 15 minewater treatment facilities (sampling sites) operating across Jharia Coalfield during the hydrological year 2019-2020 revealed variations in pH ranging from 6.5 to 8.3 with an arithmetic mean of 7.7, suggesting the nature of the coal minewater to be between mildly acidic and alkaline. The electrical conductivity (EC) of the coal minewater samples ranged between 568 and 1,389 µS cm^{-1} (mean- 1,016 µS cm^{-1}) and the TDS concentration ranged between 341 and 953 mg L^{-1} (mean- 674 mg L^{-1}). The fluctuation in concentrations of pH, TDS and EC could be attributed to existing mining conditions in the region, variations in geological formations and underlying hydrosystems (Singh et al. 2012; Rakotondrabe et al. 2018). Analysis of the ionic concentration of the minewater samples revealed calcium, magnesium (major cations), bicarbonate and sulphate (major anions) to be the dominant ions responsible for the TDS in the coal minewater of JCF, contributing 30.38, 36.93, 10.50 and 7.70% of the total TDS, respectively. Chloride (6.23%) and sodium (5.31%) were the secondary contributors, while nitrate, potassium and fluoride collectively accounted for just 1.95 % of the TDS. Bicarbonates with concentrations varying between 33 mg L^{-1} and 442 mg L^{-1}, (mean- 220 mg L^{-1}) were most dominant in the minewater of Muraidih, Shatabdi and Pootkee-Balihari Area of JCF, which could be attributed to the sequestration of CO_2 from the soil zone and reaction

of silicates in the underlying soil layer with the carbonic acid, causing carbonate dissolution. The decaying of organic matter in the subsurface of the soil zone along with root respiration results in a rise in CO_2 pressure in the subsurface environment, which on contact with percolated rainwater form bicarbonates (Singh et al. 2012). The sulphate concentration, ranged between 52 mg L^{-1} and 576 mg L^{-1} (mean- 293 mg L^{-1}) showed higher concentration in the minewater of Jamunia, Mudidih, West Mudidh, Ramkanali, Victoria, Bera and Kuya. The oxidative weathering of pyrites (FeS_2), anhydrite ($CaSO_4$) and gypsum ($CaSO_4.2H_2O$) in the underlying rocks of the region could be a major contributing factor in the higher concentrations of SO_4^{2-} (Singh et al. 2015; Tiwari et al. 2017). The spatial distribution of chloride concentration was below the maximum acceptable limit of 250 mg L^{-1} in all the sampling locations, while the fluoride concentration contributed to less than 1% of total anionic balance. Calcium concentration in the coal minewater of JCF varied between 36 mg L^{-1} and 123 mg L^{-1} (mean- 82 mg L^{-1}), contributing 44% of the total cationic concentration overall. Ramkanali (123 mg L^{-1}), Victoria (121 mg L^{-1}), Bera (113 mg L^{-1}) and Kuya (120 mg L^{-1}) showed relatively higher concentrations of calcium in comparison to the other sampling locations. This could be the result of weathering and dissolution of underlying rocks comprising of limestone ($CaCO_3$) and other calc-silicate minerals such as biotite, pyroxene, olivine, etc. (Tiwari et al. 2016). Magnesium concentration in the analysed minewater samples fluctuated between 19 and 102 mg L^{-1} (mean- 58 mg L^{-1}), accounting for 32% of total cations (TZ^+) overall, in equivalent units. West Mudidih (99 mg L^{-1}) and Jamunia (102 mg L^{-1}) showed the highest magnesium concentration among the sampled locations. The weathering of ferromagnesian minerals such as hornblende, olivine, etc. associated with metamorphic and igneous rocks and dolomite in sedimentary rocks form the primary magnesium source in the coal minewater of JCF (Tiwari et al. 2017). Among the cations, potassium was the least dominant in the coal minewater of the study area, accounting for less than 3% of the total cations.

Assessment of potability of minewater

Minewater quality of the study site for the year 2019-2020 was assessed by applying IWQI and HPI (Fig. 2) to the analyses from the aspect of major ion and trace/heavy metal concentration, respectively.

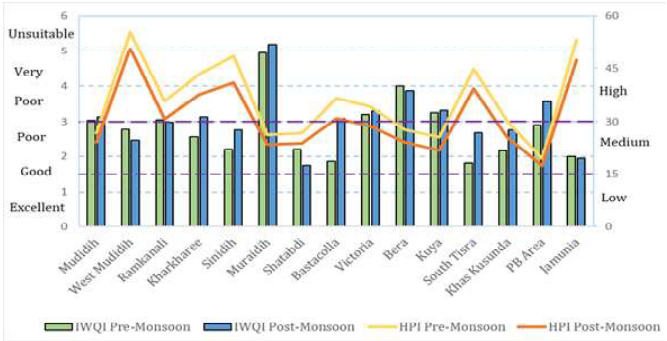

Fig. 2 Graphical representation of IWQI and HPI of coal minewater of the study area

The quality of the analysed minewater samples from their respective sampling locations were classified as- excellent (< 1), good (1–2), poor (2–3), very poor (3–5) and unsuitable (> 5); and low (< 15), medium (15-30) and high (> 30) based on the mean values of IWQI and HPI, respectively for the entire year (Table 4).

In this study, Jamunia showed high trace/heavy metal contamination (HPI- 50.30), due to high concentration of Fe (553 µg L^{-1}) and Ni (31 µg L^{-1}), while the rest of the physiological parameters including all the major ions) were within the permissible limits of drinking water standards. From this, it is apparent that installing a water softening unit or iron filters to treat the pumped-out minewater discharges to utilize it for potable purposes is warranted. Based on the results obtained by application of IWQI and HPI, respectively to the minewater analyses, it was revealed that barring the minewater of Shatabdi (1.97; 25.30), all the other locations discharged minewater that were unsuitable for direct consumption and required prior treatment (Singh et al. 2011; Singh et al. 2012; Saini et al. 2016). West Mudidh (2.63; 53.03), Kharkharee (2.85; 40.49), Sinidih (2.48; 44.76), Bastacolla (2.20; 33.75), South Tisra (2.14; 42.03) yielded minewater of poor quality with high trace/heavy metal concentration. Khas Kusunda (2.46; 27.81) yielded minewater of poor quality with medium trace/heavy metal concentration while minewater quality of Mudidih (3.08; 25.34), Bera (3.94; 25.85), Kuya (3.29; 23.71) and P.B. Area (3.24; 18.38) were deemed to be very poor with medium trace/heavy metal concentration. Ramkanali (3.02; 33.25) and Victoria (3.26; 31.53) were classified as very poor with high trace/heavy metal concentration. Muraidih (5.08; 24.78) was found to have a high IWQI value and thus deemed unsuitable for direct consumption. Although the pumped-out coal minewater of Jamunia had a IWQI value of 1.97, it requires treatment prior to consumption due to high trace/heavy

Table 4: Minewater quality classification of the study area for the year 2019-2020

Sampling location	Mean IWQI value	Mean HPI value	Category of minewater quality
Mudidih	3.08	25.35	Very poor, medium trace/heavy metal contamination
West Mudidih	2.63	53.05	Poor, high trace/heavy metal contamination
Ramkanali	3.02	33.25	Very poor, high trace/heavy metal contamination
Kharkharee	2.85	40.50	Poor, high trace/heavy metal contamination
Sinidih	2.48	44.75	Poor, high trace/heavy metal contamination
Muraidih	5.08	24.80	Unsuitable, medium trace/heavy metal contamination
Shatabdi	1.97	25.30	Good, medium trace/heavy metal contamination
Bastacolla	2.48	33.75	Poor, high heavy metal contamination
Victoria	3.26	31.55	Very poor, high trace/heavy metal contamination
Bera	3.94	25.85	Very poor, medium trace/heavy metal contamination
Kuya	3.29	23.70	Very poor, medium trace/heavy metal contamination
South Tisra	2.25	42.05	Poor, high trace/heavy metal contamination
Khas Kusunda	2.47	27.80	Poor, medium trace/heavy metal contamination
PB Area	3.24	18.40	Very poor, medium trace/heavy metal contamination
Jamunia	1.97	50.30	Good, high trace/heavy metal contamination

metal concentration (HPI- 50.30). The mean HPI values of trace/heavy metal concentrations were found to be 35.62 for the pre-monsoon month of March-April and 31.09 for the post-monsoon month of October-November, which fall below the critical pollution index limit of 100. Minewater quality during post-monsoon was found to be better in terms of trace/heavy metal concentration (Panigrahy at al. 2015; Singh and Kamal 2017; Chaturvedi at al. 2018).

The integrated approach of combining both IWQI and HPI for qualitative studies of minewater could prove to be efficient in characterization and classification of water for its utilization in various purposes. This method helps in making quick decisions about the destination of pumped-out minewater discharges and its subsequent treatment (if any) to improve the quality of the water.

CONCLUSION

The integrated approach for water quality assessment checks the limitations of eclipsing, aggregation and ambiguity posed by traditional water quality indices, considering the maintenance of ionic balance in potable water, and thus representing optimal water quality as per the standard guidelines with respect to human health. The integrated model incorporates percent deficit in its calculations which makes the users aware of the pollution levels reaching the threshold limit, giving enough time to address the situation, thus, rendering it flexible in its usage as the percent deficit can be adjusted as per the nature of the environment. This would facilitate quicker operational responses to different types of pollution events and protect the water plant installations. This attribute would also allow for a wide applicability across different regions around the world. Water management strategies could be greatly helped by the integrated approach as it would quick decision making about the destination of the pumped-out coal minewater or the type of treatment to be employed to improve the quality of the minewater. Principal Component Analysis (PCA) of minewater quality is an efficient chemometric tool to explore the relationship between physicochemical water quality parameters and the geological domain. In the current study, PCA revealed the trace/ heavy metal concentration in pumped-out underground minewater to be from a mixed source of anthropogenic and lithogenic origin, highlighting the impact of mining activities on groundwater. The results of the study suggest that multivariate statistical processes using PCA, and HCA could be used to improve water monitoring

strategies and management of water resources. The multivariate statistical approach holistically identifies areas of contamination, giving an assessment of the quality of minewater and its suitability for potable purposes and laying the framework for future investigations in augmenting minewater for potable purposes.

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Coal Quality and Its Utilization –Need

Aarif Jamal* Prashant Modi**

ABSTRACT

Coal industry is important for developing countries like India. India has a vast reserve of coal in the world, come on the world's fifth-largest coal proved reserves including Lignite (Approx. 100 billion tones). Coal is a mixture of heterogeneous rock materials, including organic as well as inorganic contents in it. The most advanced technology is used for coal quality characterization to meet as per the demand in the coal based industries. In the thermal power industry , coal quality is being assessed by the Proximate analysis and Gross Calorific Value (GCV). A case study has been discussed in this paper about coal quality, its present utilization and possibility for other uses. . The average value of moisture % is 2.8, Ash% is 15.12, Volatile matter% is 26.71 and Fixed carbon% is 55.32 in 123 coal samples. Relation of ash % in coal and its grade is inversely proportional to each other. There is demand for low ash coal both by producer and consumers also. By the peaks in the XRD graph, the minerals present in coal has also studied. This study has helped to understand the reason of grade variation in coals, which will be helpful in thermal industries. The inorganic content (minerals) and sulphur amount in coal indicate the allocthonous deposition in Mahanadi Basin of the Gondwana group.

INTRODUCTION

Coal is playing a very crucial role in developing countries like India because coal is very cheap and easy to use. India has a vast reserve of coal. The Bulk of the coal produced in India is thermal coal (Non-cooking coal). It provides 60% of the nation's electricity (Joonseok and Johannes, 2019). The major coal-bearing formation of the Gondwana Basins is the Lower Permian Barakar Formation (Mukhopadhyay, 2018; Dutta, 2002). The coal production reached up to 8000 MT in 2019 worldwide. This amount of production in world tells us the dependence on coal till now and in future also. Coal in India has been mined since 1774, and India is the second largest producer and consumer of coal after China, mining 783 million metric tons in 2019. Coal supplies over 50% of energy in India. Around 30% of coal is imported. Due to high demand and poor average quality, India imports coal to meet the requirements of its steel and power plants.Most of coal is burned to generate electricity.

Coal is an organic-sedimentary rock, contains organic and inorganic content. Due to heterogeneous nature of coal, many analytical techniques were used for the characterisation of coal. In recent years, many advanced techniques have come, which were used in the sophisticated analysis. This sophisticated analysis helps to understand more about coal and its characterisation. Coal quality details must be studied in order for its relation with carbon and the other parameters. The study of coal quality principally gives knowledge about moisture, ash, volatile matter, mineral matter, Carbon, Hydrogen, Nitrogen, Sulfur, Oxygen, trace elements, useful-heat value, and calorific value in coal. This all data, in turn, produce data on the technological performance of coal.

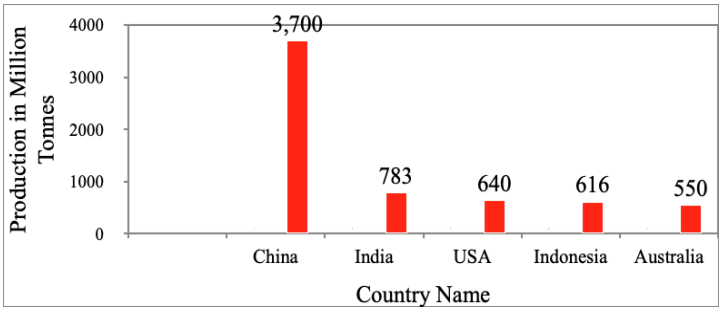


Fig. 1: Coal production by top 5 Countries globally

Figure 2 : Photograph of Coal seam of an Opencast Mine

**Professor **Research Scholar*
Department of Mining Engineering, Indian Institute of Technology (BHU), Varanasi, India-221005

MATERIALS AND METHODS

The research work was begins with a constituent of extensive field work in the study area. The coal samples were collected working seam faces of an open cast mine. The samples were collected by following pillar sampling method. Every collected coal samples were put in Ziplock plastic bags and sealed immediately carefully to minimise oxidation and further any kind of contamination. Samples were brought tolaboratory for megascopic characterisation in hand specimen. After this charcaterization, samples were crushed according to the mesh size required for analysis in the laboratory.Further, these samples were also used for the Geochemical analysis for the determination of their Proximate analysis and Ultimate analysis constituents. With the help of Proximate analysis, we can determine the Moisture, Ash, Volatile matter and Fixed carbon percentage in coal samples. Ultimate analysis of samples determined the Carbon, Hydrogen, Nitrogen, Oxygen and Sulphur percentage in samples.

The proximate analysis was carried out in Mineral processing and beneficiation laboratory of Mining engineering department, I.I.T., B.H.U. Varanasi. Some samples also carried out by Thermogravimetric Analysis (TGA) in Central instrument facility of I.I.T., B.H.U. Varanasi. Proximate analysis evaluates moisture content, ash yield, volatile matter and fixed carbon in weight percent. The analysis was performed by taking -72 mesh size coal samples. The proximate analysis was done as per specifications laid down by Bureau of Indian Standards (BIS, 2003). The results of proximate analysis constituents were used for assessing the grade of coal. Ultimate analysis was carried out in M.P. Singh Laboratory of Coal and Organic Petrology, Centre of Advanced Study in Geology, Banaras Hindu University, Varanasi, Uttar Pradesh, India. The ultimate study is performed on Elemental analyser. Gross Calorific Value is described as per ISO standards as measuring heat value - absolute value of specific energy of combustion for unit mass of a solid fuel burned in oxygen in a calorimeter bomb under specific conditions. The test is carried out on air dried sample and is calculated to as received and dry basis. Bomb calorimeter was used Parr 6100 model, present in Mining engineering department. Proximate analysis by Thermogravimetric Analysis (TGA) was carried out in Central instrument facility of I.I.T. (B.H.U.), Varanasi, India for the samples (-72 mesh size).

RESULTS AND DISCUSSIONS

Geochemical characterization is an important part of study for coal samples. In geochemical characterization, we determined the Proximate analysis, Ultimate analysis and gross calorific value.

PROXIMATE ANALYSIS

The proximate analysis of coal from table 1 revealed the variation of value among the samples. Moisture% ranges from 1.95-3.52, Ash% ranges from 12.50-20.60, Volatile Matter% ranges from 15.98-32.62 and Fixed Carbon% content is calculatedby subtraction the sum of moisture, volatile matter, and ash contents from 100, ranges from 47.45-69.55. Less ash contents of these coals shows the coal are of steel grade & washery grade if coking property is present in coal samples. Gross Calorific value of coal samples is from Grade G9 to G6.

ULTIMATE ANALYSIS

Ultimate analysis coal reveals the elemental distribution in among coal samples. Carbon percentage in coal samples ranges from 59 to 65%, Hydrogen ranges from 4.1 to 4.79%, Nitrogen ranges in 0.5%, oxygen ranges from 29 to 34% and Sulphur ranges from 0.5 to 1.7%. Sulphur content in coal samples are ranging from 0.54 to 1.76% that is also favours the allocthonous deposition of coal in basin.

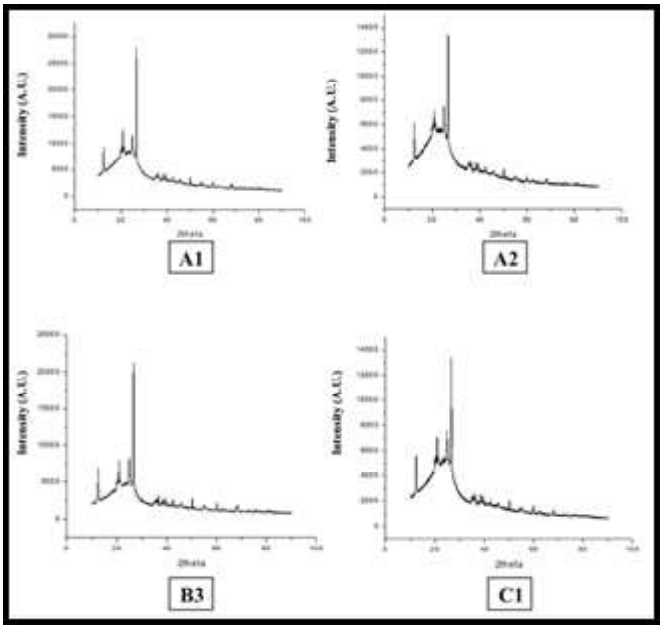
Sample	M	A	VM	FC*	C	H	N	O	S	H/C	O/C	GCV(Kcal/gm)
A1	3.52	13.7	32.62	50.08	64.11	4.45	0.59	15.3	1.76	0.8	0.1	5537
A2	1.95	12.5	15.98	69.55	63.38	4.79	0.65	11.4	1.10	0.8	0.1	5638
B3	2.34	20.6	29.60	47.45	59.13	4.1	0.57	14.4	1.21	0.8	0.2	4791
C1	3.41	13.7	28.65	54.21	65.36	4.2	0.54	26.7	0.56	0.9	0.3	5798

*by the difference

Table no. 1 Proximate analysis, ultimate analysis result & gross calorific value of coal samples

X-RAY DIFFRACTION

X-ray diffraction analysis of coal samples, graph were plotted by the Origin software. In graph different peak shows the different minerals presence in coal samples. Graph is plotted between the Intensity (A.U. = Arbitrary Unit) on Y-axis & 2theta degree (2?)values on X-axis. According to XRD data, mineral phase identified by the XRD were Quartz, Kaolinite, Feldspathoid, Anatase(Rutile), Aditionally, peaks of Iron Oxide, Birnessite, Graphite, Beryllium Bis(Hypophosphate), Sodalite,Cr-Oxide, Cossaite,



Caesium-Element), Si-Carbide, Millosevichite, Spodumene(Pyroxene), Al-Phosphate, Muscovite, Spangolite were also detected incoal samples. There are few minerals which were reported in coal samples of SECL. Mineral groups present in samples are Native elements, Phosphates, Silicates, Oxides & Sulphates etc. This is the main contribution for mineral matter in form of ash in coals.

Mode of formations of these samples in coal are :

1. Terrigenousdetritalminerals
2. Authigenicminerals
3. Diageneticalterations
4. Plantoriginmaterials

NEED FOR TRANSITION

Global climate change issues are forcing the coal producing countries not to burn coal. Coal production is not an issues but coal burning should be discourage to reduce carbon emission. Coal consuming countries are under pressure to phase out coal by 2050. Phasing out coal based power is also one of the major targets of climatic agreements. At the Paris Agreement in 2015,the world agreed to make efforts to limit the increase in global average temperature to well below 2 degree Celsius above pre-industrial levels and to try its best to limit the increase to 1.5 degree Celsius .

There is need to transition to go for alternative source of power and simultaneously utilization of coal without burning. It has been estimated that to meet the 1.5 degree Celsius climate targets, coal share in global energy supply should decline between 73-97 %by 2050.This leads to elimination of coal industry including mining operation.

CONCLUSION

Coal from open cast mine is suitable for thermal grade coal ranges from G9-G6 grade of coal according to ministry of coal, Government of India. The coals are also suitable for steel grade-I to Washery grade-I if coking properties present due to it's ash content. In the coal samples we can see that the sample B3 has a lesser gross calorific value (GCV) than other coal samples, also can be correlated to a high O/C ratio relatively among other coal samples. H/C ratio decreases with the degree of coalification and can be seen in coal samples of B3 that is lesser. Coal is a bituminous rank of coal. The minerals found in coals are because of the allochthonous nature of coal and sulphur range in coal samples also proves it. XRD showed the presence of inorganic content of coal which plays an important role to determine the rank and grade of coal. This inorganic content is biggest reason for lowering the grade of coal and will lead to burden over producers and consumers in costing of coal, which is required to be cut for profitable extraction. This inorganic matter should be removed after mining through the beneficiation (washing) of coal, so that in transportation from mining areas to site of thermal industries it will be less burden.

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Development of Rock Mechanics Software for Application in Mines

Dr. G.G. Manekar*

ABSTRACT

All the mining operations for exploitation of ore have moving fronts either in opencast or in underground workings. Moreover, the rocks are non-homogenous in nature. It is therefore mandatory to know the various rock mechanics parameters during mine planning, design and its application during mining operation of the mine.

The main function in opencast working is to ultimate pit slope design, blasting parameters for desired fragmentation and dump stability and many more. Excavation width, height and type of support and support density for reinforcement of underground structures for safety and stability of underground mine.

Many attempts have been made worldwide for establishment of standards for safety. In this context, after the VI safety conference in India, the regulation authority DGMS has imposed the establishment of RMR for support in coal and non-coal mining operations in the year 1986. Thereafter establishment of NIRM at KGF, Barton's Q has been introduced for support requirement. The journey from various parameters like Deere's RQD to Barton's Q and its application is reproduced for mining and geologist for implementation in the operations for higher safety standards and better productivity of mining operations.

The knowledge and skill of mining operations of weak, varying dip directions, narrow width mobile manganese ore body of meta – sedimentary deposit of Sausar belt of central India, We have developed software in association with MOIL Limited, in which all the parameters required for derivation of RMR and Q have been incorporated. The beauty of the software is that there is no Human-Machine Interference (HMI) and output is directly placed on the graph Just in Time (JIT). In a day researcher could analyse many samples for standardisation of its application considering the actual variations at the site for safety and productivity. We are developing the software for many more applications by using artificial intelligence (AI) platform in various mining operations.

Keywords: Cable Bolting, RMR, Barton's Q, Roof Bolts, Stowing

INTRODUCTION

The software is being developed in association with MOIL and many attempts were made to confirm its application. The geological formations of the mining lease area in central India belong to munsar formation of sausar group. The sausar fold belt is an important Mesoproterozoic (~1300 Ma age of Sausar Group) mobile belt in the Central India Peninsular Shield. Sausar belt extends for a length of over 215 km from Ramakona to Paraswada (Baihar) with an average width of 35 km and covers an area of about 7000 km² in the state of Maharashtra and Madhya Pradesh. It is an arcuate fold belt trending ENE-WSW with convexity to the south (Ramakrishna M and Vaidyanandhan, 2010).

**Director, Ahinsa Enterprises, MSME Registered – UDYAM – MH-20-0047717, Consultancy and Development Trading, 311, Dewanjalee Apartments, Reshimbag, Nagpur – 440024*
Corresponding Author: ggmanekar61@gmail.com

GEOLOGY OF AREA

The land based deposits are mainly of three types; (i) Hydrothermal Mn deposits; (ii) Sedimentary Mn deposits; and (iii) Lateritoid Mn deposits, of which second group is most important economically (GSI – 2018). The manganese deposits in the area are associated with rock of Sausar. These rocks are mainly meta-sediments composed of quartzites, various types of schist and gneiss. These are found at the base of the lower most Sausar Series of rock and have been involved in the movements along with the other rocks and thus have developed certain features which make it difficult to identify them from the other rock of the Sausar Series. The ore bed in this area occurs in the Munsar formation of the Sausar rock belongs to the Dharwar metasediments and comprises of "various types of schists and gneisses, dolomitic marble" calgranulites and Biotite gneiss is found at the base of the lower most Sausar formation and it is involved in the movement with rocks of the same group. During this movement it might have developed some

features which make it difficult to identify. Moreover, these rocks have undergone a high degree of metamorphism and the rocks are characteristically metamorphosed.

ROCK MASS CLASSIFICATION

The rock mass classification parameters namely, rock quality designation (RQD), joint set number (Jn), joint roughness number (Jr), joint alteration number (Ja), joint water reduction factor (Jw), stress reduction factor (SRF), uniaxial compressive strength (UCS), spacing of discontinuities, joints conditions, orientation of discontinuities and hydro-geological conditions, were estimated for ore as well as wall rocks using 3-D geological mapping and core logging. Based on these parameters, the rock mass has been characterized using Q-system (Barton, 1976) and RMR system (Bieniawski, 1973). Empirical estimation of support requirement for the modified stope design was made using the above two rockmass classification systems. It is observed that the foot wall and hang wall rocks have Q value of 4.75-9.37 and 6.06 - 9.38 respectively (Raju G D, NIRM, 2016), categorizing them both as “Fair”. On the other hand, the estimated “Q” value for the Ore is 22.5 - 66.67, which falls under ‘Good’ to ‘Very Good’ category as per the Q chart. Another widely used rockmass classification system, RMR is also examined. It is observed that the average RMR values of footwall, hang wall and ore body is estimated to be 44.5, 59 and 56 respectively (CMRI, 2001). Obtained Q and RMR value is presented in Table 1.

Table 1: Obtained Q Value and RMR of Munsar Mine

Geo-technical Parameters of wall rocks and ore body for Q-system		
Location	Rock description	Q-Value
Hangwall	Mica schist/Quartz mica schist/Rhodonite with mica schist	6.06 – 9.38, (Fair)
Ore zone	Mn-ore body	22.5 – 66.67 - (Good – Very Good)
Footwall	Quartz mica schist/Mica schist	4.75 – 9.37, (Fair)
Geo-technical Parameters of wall rocks and ore body for RMR		
Location	Rock description	RMR
Hangwall	Mica schist/Quartz mica schist/Rhodonite with mica schist	59, (Fair)
Ore zone	Mn-ore body	51 – 61, (Fair – Good)
Footwall	Quartz mica schist/Mica schist	33 – 56, (Poor – Fair)

EMPIRICAL DESIGN

The maximum stope width of 7 m is considered for the modified stope design for the mine. The excavation support ratio (ESR) is taken as 1.6. It can be observed from the ‘Q’ chart, which is shown in Figure 1 that the region falls in unsupported area for 6 m stope width. However, considering the damage due to blasting and other unforeseen geological effects, systematic bolting is found correct for new stope application in which the haulage road and cross cut is placed in the foot wall rock.

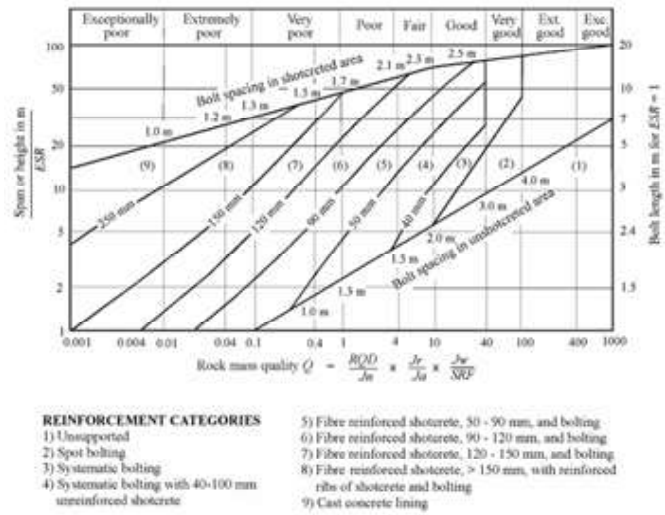

Fig.1: The excavation support ratio (ESR) as per the ‘Q’ chart

Based on the RMR chart proposed by Bieniaski (1973) for the span, a stand-up time, it has been examined for the RMR 56 of ore body. It is found that the stand up time for an average stope width of 7 m is between one week and one month. However, the stope is supported with 1.5 m long rock bolts and hence the actual stand up time for the stope width will be much higher than that shown in RMR chart.

ROCK MASS CLASSIFICATION

The rock mass classification parameters namely, rock quality designation (RQD), joint set number (Jn), joint roughness number (Jr), joint alteration number (Ja), joint water reduction factor (Jw), stress reduction factor (SRF), uniaxial compressive strength (UCS), spacing of discontinuities, joints conditions, orientation of discontinuities and hydro-geological conditions, were estimated for ore as well as wall rocks using 3-D geological mapping and core logging. Based on these parameters, the rock mass has been characterized using Q-system (Barton, 1976) and RMR system (Bieniawski, 1973). Empirical estimation of support requirement for the modified stope design was made using the above two rockmass classification systems. It is observed that the foot wall and hang wall rocks have Q value of 4.75-9.37 and 6.06 - 9.38 respectively (Raju G D, NIRM, 2016),

categorizing them both as “Fair”. On the other hand, the estimated “Q” value for the Ore is 22.5 - 66.67, which falls under ‘Good’ to ‘Very Good’ category as per the Q chart. Another widely used rockmass classification system, RMR is also examined. It is observed that the average RMR values of footwall, hang wall and ore body is estimated to be 44.5, 59 and 56 respectively (CMRI, 2001).

In association with MOIL, we have developed in house “A system and a method for rock mass characterization and rock support system in mining” and filed a patent on 31.01.2020 and published on 14.08.2020 for examination by patent office. The app in which all the parameters can be put directly and software indicates the actual width of opening and stand up time in RMR and required support system in Barton’s Q. There is no human-machine interference in this software. This software/app has been developed in JAVA programming language with JAVA development kit (JDK) version 1.8.0_u131 or higher and Front End in JavaFX using Netbeans IDE. It saves times and accurately furnishing the opening width of excavation, stands up time and support system. Software generated RMR value and Q is presented below in Fig. 2 and 3.

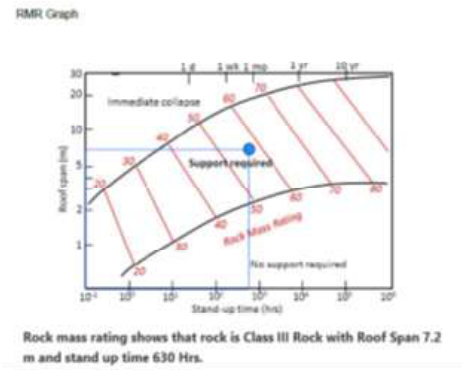

Fig. 2: RMR Graph

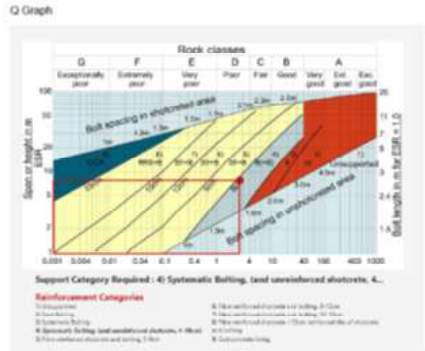

Fig.3: Graph of Barton’s Q

EMPIRICAL DESIGN

The maximum stope width of 7 m is considered for the modified stope design for the mine. The excavation support ratio (ESR) is taken as 1.6. Based on the RMR chart proposed by Bieniaski (1973) for the span, a stand-up time, it has been examined for the RMR 56 of ore body. It is found that the stand up time for an average stope width of 7 m is between one week and one month. However, the stope is supported with 1.5 m long rock bolts and hence the actual stand up time for the stope width will be much higher than that shown in RMR chart.

CONCUSIONS AND RESULTS

The software instantaneously gives the RMR which can evaluate stope width and safe time for support and Barton’s Q gives the type of support and density. Moreover, the data incorporate is available with input parameters in data sheet. It is a beauty of the software that there is no Human-Machine-interference (HMI) and the calculate RMR and Q is placed in graph directly. The attempts are going on to use the software in Artificial Interference (AI) so that various functions like blast design, bench slope, ultimate pit slope can be evaluated with this software.

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